# MINING GEOLOGY

1948

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Vol. 178
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# TRANSACTIONS

OF THE

# AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS

(INCORPORATED)

## Volume 178

American Institute of Mining, Metallurgical and Petroleum Engineers. //

## MINING GEOLOGY
## 1948

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PAPERS AND DISCUSSIONS PRESENTED AT MEETINGS HELD AT NEW YORK: FEB. 9–12, 1942;
FEB. 15–18, 1943; FEB. 20–24, 1944; MARCH 17–19, 1947; FEB. 15–19, 1948; AT
CHICAGO, FEB. 25–28, 1946; AT TUCSON, OCT. 28–30, 1946;
AT SAN FRANCISCO, FEB. 14–17, 1949.

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PUBLISHED BY THE INSTITUTE
AT THE OFFICE OF THE SECRETARY
29 WEST 39TH STREET
NEW YORK 18, N. Y.

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PRINTED IN THE UNITED STATES OF AMERICA

THE MAPLE PRESS COMPANY, YORK, PA.

# FOREWORD

The art of mining geology has two major aspects: (1) the determination of the disposition of ores in the crust of the earth, and (2) the assisting of mining engineers in exploiting ore deposits in such manner as to make the largest profit. Perhaps at the moment the first phase is most important—finding new ores and new ore deposits to redress the shrinkage in known reserves that resulted from the tremendously high rates of production since 1941 and the coincidental slowing down in exploration and development.

"New" ore can be found in any one of three sorts of localities: areas in and around known mines; areas that have favorable surface indication but that hitherto have been unproductive; and areas that are devoid of favorable surface indication but that the geologist, endowed with imagination, can inferentially choose as worthy of intensive study and examination.

Differences of opinion may—in fact do—exist as to the quantity of exploitable ores that eventually will be found in relation to the quantity that has been found and mined in the past; but there can be no justifiable differences of opinion as to the necessity for conducting the most intensive search for new ore. Geologists will seek the aid of geophysicists in enterprise of this kind. Such search will cost money—lots of money; but it is safe to say that the money will be forthcoming.

New and better instruments, apparatus and devices will be developed, but the most important factor will be human ingenuity and resourcefulness. Geologic art will become more and more important; the geologist will be a key figure in the search for exploitable mineral deposits.

Progress in mining geology as in any other kind of applied science or technology is based, in no small degree, on authoritative written records of the investigations, findings and performance of many practitioners. The present volume consisting of 38 special articles constitutes such a record. An examination of the contents pages will show the wide range of subjects covered and the high standing in his particular field of each author. To all of these and particularly to D. H. McLaughlin, H. E. McKinstry, Carlton D. Hulin, and Philip J. Shenon, who successively have held the post of Chairman of the Committee on Mining Geology since 1941, the Institute and the profession as a whole are deeply indebted.

A. B. Parsons

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# Wall Rock Alteration at Butte, Montana

BY RENO H. SALES* AND CHARLES MEYER,* MEMBERS AIME

(New York Meeting, February 1948)

## ABSTRACT

AT Butte, successive zones of sericitized and argillized quartz monzonite occur around every ore-bearing fracture regardless of its size, attitude, or relative age. The two types of alteration always occupy the same relative positions; sericite adjacent to the ore-bearing vein, clay minerals always between the sericitized rock and fresh quartz monzonite, except where overlap of alteration effects between adjacent fissures has eliminated the lower grade mineralogical products, as is the case in much of Butte's Central zone of pervasive sericitization. As long as active circulation continued in the channel, each zone migrated away from the fissure, that is, it grew at its outer edge and simultaneously receded at its veinward edge because of encroachment by the next innermost zone. The different mineralogical and chemical responses within the wall rocks to the attack by the ore fluid are dependent not on a drastic about-face change of composition of the fluid in the channel relative to its ability to alter the quartz monzonite but on continuously varying conditions of physicochemical environment within the wall rock outwardly from the vein.

## INTRODUCTION

From the economic viewpoint it is highly desirable that much effort be expended for the purpose of increasing our knowledge of ores and their origin. It has been the observation of the senior writer, many times expressed, that the mining companies have spent too little money in research on the geology of nature's deposits of copper, lead, and zinc, which we know to be of limited extent and which, when exhausted, must be replaced by new discoveries.

Experience at Butte and elsewhere has forced the conclusion that observation and mapping in the field must be supplemented by laboratory work if we are to arrive at a reasonably correct understanding of the geology of ore bodies and their environments. It is only through the medium of such correct understanding of geology that we shall be able to accomplish the maximum toward aiding mine operations and at the same time provide a proper background for field examination work whether it be geological or geophysical.

The Anaconda Copper Mining Company's answer to the need above expressed has been to equip a geological laboratory at Butte. The reason why Butte was selected is obvious, for situated there is one of the world's largest and geologically most interesting concentrations of copper and zinc. Furthermore, because of the great extent of underground workings both laterally and vertically, the opportunities for observation and study of ore occurrences probably are unequaled anywhere.

The laboratory was established with five principal objectives in view. These were:

1. As an aid to Anaconda Company mine operations at Butte and elsewhere, this work to supplement that of the mine geologists.

2. To aid in acquiring as complete an understanding as possible of the behavior of Butte veins and ore bodies laterally and in depth.

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Manuscript received at the office of the Institute April 5, 1948. Issued as TP 2400 in MINING TECHNOLOGY, May 1948.

* Chief Geologist and Research Geologist, respectively, Anaconda Copper Mining Co., Butte, Mont.

3. To render assistance to the metallurgist on problems of ore concentration.

4. To make rock, mineral and ore determinations for field geologists employed on examination work.

5. To contribute to the existing knowledge of ore deposits.

The work thus far has been concentrated mainly on the important Butte problem. It is obvious that any attempt made in a laboratory, or out of it, to attain objective number two, that is, determining the behavior of an ore body in depth beyond where it can be observed, will meet with many difficulties.

From the viewpoint of procedure there seemed to be two available points of attack on the Butte problem: (1) the vein minerals comprising the veins, and (2) the ever present wall rock alteration in association with the veins and fissures. Rock alteration was selected for the first line of attack for the reason that a study of wall rock effects might afford a clue as to the nature of the ore-bearing solutions passing a given point within a vein, at which point certain types or groups of minerals had been deposited. In other words a clue might be provided that would aid in the solution of the problems of mineral zoning in the Butte district.

Before offering some of the results of this rock-alteration study, it might be well to state that the geology of Butte was the subject of an extended investigation by W. H. Weed[1] during the years from 1901 to 1906. The results of his work were published in 1912, and in the same year a paper by Charles T. Kirk[2] appeared on mineralization conditions in the copper veins at Butte. A descriptive paper[3] on ore deposits at Butte was presented by the present senior author at the Butte Meeting of the AIME in 1913. A number of descriptions of sulphide relationships in the Butte veins appeared shortly thereafter, but in more recent years little concerning the geology of the district has been published.

Since 1913 there have been very extensive mine workings made in the Butte district both laterally and vertically. The present deepest mining level is the 4200 as compared with the 2800 in 1913. Around the fringes of the copper zone large zinc-ore developments have taken place, and in addition a substantial tonnage of manganese ore has been produced from the Emma mine. Manganese ore (rhodochrosite) was not recognized as an ore possibility in 1913.

As a reminder, it is in order to give a very brief résumé of Butte geology. Butte is an extensively fissured area. The country rock is quartz monzonite into which have been intruded a few narrow dikes of quartz porphyry. A later rhyolite breaking up through the western part of the district is generally thought to be post-vein in age.

The quartz porphyry was followed by extensive fracturing and mineralization resulting in what is known as the East-West system of veins. These veins are by far the most extensive and persistent of any in the district. Next, named in order of appearance, are the Blue faulting fissures, also heavily mineralized, and the northeast fissures, only slightly mineralized. Still later came the faults of Rarus and Middle ages, which cut and displaced all veins and fissures of the earlier systems. No ore bodies have been found in these later faults, yet the presence of wall rock alteration along them marks the time of their appearance as prior to the time mineralizing action had ceased entirely.

The ore and gangue mineral distribution within the veins has been previously described. They exhibit a broad mineral zoning within individual veins as well as within vein groups, arranged radially upward and outward from an undisclosed deeper central source. Nearest the source are the copper minerals; next above a mixture of zinc and copper sulphides, including chalcopyrite; then zinc and lead sulphides with manganese as carbonate or silicate; then quartz with very limited sphalerite, galena, chalcopyrite, and manganese. Yet outward from this economic type of mineralization is quartz and lime-rich carbonate with very little scattered pyrite.

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[1] References are at the end of the paper.

Since 1913, deeper mine workings have greatly expanded the areal extent of the so-called "Central" area of quartz monzonite alteration. Increasing widths of altered quartz monzonite along vein walls with greater depth is the rule. Also, in certain parts of the district away from the Central zone, deep level workings have disclosed substantial areas of rock alteration, whereas but little or only a moderate thickness of alteration is present along vein walls vertically above at higher levels.

From the economic point of view the Butte problem involves the important question of the persistence of copper and zinc ores in depth. Already some reasons have been given why the study of rock alteration was undertaken as the first line of attack. Also, the idea had been expressed as to whether a full knowledge of the type and nature of Butte wall rock alteration in the higher levels, where many mineralized fissures are barren of copper, might enable a more confident prediction to be made in advance of exploration as to the presence of copper in the deeper portions of those veins. This, of course, was a big order, but the thought supplied a strong incentive to proceed with the study. Another important objective was to determine, if possible, what bearing, if any, a given type of alteration had on ore continuity at successively deeper levels.

We should add that in our attack on the problem of rock alteration we are not alone, for in recent years a number of geologists have given much time to the subject in the field and in the laboratory. Some of these investigations have proceeded on the broad theory that rock alteration might give clues that would prove helpful in the search for ores. These objectives, in effect, spelled attempts to determine something of the nature and composition of the ore-bearing solutions ascending from the depths. It is obvious that a sound theory of ore deposition must rest upon a foundation fashioned from facts. That carefully laid foundation should include a complete knowledge of the time relationships involved in the structural history of any area under consideration. It must incorporate also detailed information concerning the identity, stability ranges, and interrelationships of the minerals involved. Errors of improper delineation of mineralogical stability ranges, or inadequate consideration of all the possibilities, may become greatly magnified in the finished theory, sometimes to the extent of invalidating the fundamental concepts of that theory. Interpretations about the white mica group of minerals, for example, may be cited as a case in point. Present knowledge of mineralogical stability ranges, though expanding rapidly thanks to many investigators concerned with the problems of mineral syntheses, is still very much of the permissive rather than of the compelling type. In fact, even the identity of several qualitatively and quantitatively important groups of alteration minerals is not adequately known.

These are mineralogical problems, therefore an investigation calculated to delineate a sound theory of ore deposition for a given deposit must begin with an intensive mineralogical study. The Butte laboratory is trying to meet this requirement and much time has been devoted to the study of minerals and mineral variations involved in the alteration of the wall rocks of Butte veins and fissures.

At the very outset it became necessary to choose a method of attack; in other words, to devise a coordinated field-laboratory technique. As already stated, the wall rocks of the veins and fissures of all ages exhibit various shades and degrees of intensity of alteration effected by mineralizing solutions. Obviously, alteration effects will be overlapping, or superimposed, wherever veins of different ages are closely spaced, in areas of fissure intersection, and where there has been fracturing or re-opening of a mineral-filled vein permitting a renewal of solution movements within or along the vein. In the so-called Central area it is extremely rare to find a piece of quartz monzonite or quartz porphyry not exhibiting intense alteration.

Selecting samples from that extensively altered area, or at random from vein walls at various elevations in different parts of the district, did not appear to be a sound procedure where superimposed alteration effects must be avoided to the greatest extent possible. It was thought highly desirable that the samples be chosen so as to provide a sound basis for: (1) a record of the results of continuity of action of a solution on the wall rock outward from the vein or fissure, and (2) for establishing the true nature of the gradational changes concurrently taking place in the minerals outward from any given point along the walls of the vein. And, conversely, it seemed that only from the results of a determination of the point-to-point chemical and mineralogical effects outwardly from the vein to fresh rock, or inwardly from fresh rock to the vein, all interpreted in the light of the complex structural pattern at Butte, would it be possible to arrive even at tentative conclusions such as are set forth later in this paper regarding the nature of the solutions effecting the alteration.

To meet these requirements it was necessary to select for sampling and study bands of alteration believed to have escaped the action of solutions passing through fissures other than the one directly under consideration. Samples were chosen representative of the alteration effects at various elevations in all parts of the district, and from wall rocks enclosing veins of different ages and different mineral compositions. More details of these procedures will appear in a later section of this paper.

And finally, because of the close genetic interrelationship between wall rock alteration and mineral deposition in and along fissures, the story of Butte rock alteration cannot be completed until a full understanding is gained of the age relations of the vein minerals and their distribution in vertical and horizontal range throughout the district. The study of rock alteration will be continued with increasing emphasis on the relationship between the process of sulphide precipitation and the chemistry of alteration.

## DISTRIBUTION OF ALTERATION PRODUCTS

In the areas immediately surrounding the so-called "Central" zone of the Butte District the mineralized fissures are large persistent structures separated by relatively wide zones of subordinately fractured quartz monzonite. Within and along these major veins the wall rock is profoundly altered mineralogically and chemically. But between such major alteration envelopes most of the rock is fresh, showing alteration only where it is traversed by subsidiary channels. Each of these smaller faults or joints is bordered by its own alteration envelope—a complete miniature, in qualitative mineralogy, of the larger ones. Hence, a relatively small scale example (Fig 1) is adequate as an illustration for a preliminary definition of the mineralogical identity and spatial distribution of the alteration products adjacent to a mineralized fracture at Butte.

The alteration minerals are arranged in zones parallel to the fracture. These zones may be named according to their mineralogical content since by this criterion they are readily recognizable even megascopically. Beginning with the vein itself and proceeding outward toward unaltered quartz monzonite, one encounters successively zone A (Fig 1), the sericitized zone, in which the predominant hydrothermal mineral is a white mica; and zone B, where clay minerals, principally of the kaolinite and montmorillonite groups, are dominant. In published accounts of other deposits the useful term "argillized zone," proposed originally by T. S. Lovering[4], has been applied to units of hypogene alteration similar to zone B.

In Fig 1 it is apparent that the white, bleached, veinward portion of the argillized zone, which is relatively rich in kaolinite, is easily distinguishable from the outer greenish part of that zone where montmorillonite-type clay is more abundant. These two phases of the argillized zone are actually intergradational in transition, yet for descriptive purposes it will be useful to refer to them independently as the kaolinite and montmorillonite subzones. Local areas of silicification in the sericitized zone merit similar evaluation.

All of these mineralogical zones are controlled in their identity and in their relative positions by the channel which now holds the vein. They are arranged in good bilateral symmetry around the fracture when the latter is simple and small and cuts otherwise unbroken quartz monzonite. Or, contrarily, they may be deformed in attitude and disposal to the extent that the fracture is intersected by other channels, or when it encounters a change in wall rock type. But even in such a maze of complexity of intersection as is the rule at Butte, considering each vein and its attending alteration suite independently, the same qualitative mineralogy prevails in the same relative distribution regardless

FIG 1—DISTRIBUTION OF ALTERATION PRODUCTS ADJACENT TO A VEINLET.
A. Sericitized zone. B. Argillized zone. C. Fresh quartz monzonite.

The argillized zone here shows three phases: the white kaolinite-rich portion in contact with the sericitized zone, the dark greenish band of montmorillonite-rich and nontronite plus allophane-hisingerite-bearing plagioclase and hornblende alteration adjacent to fresh quartz monzonite, and the lighter green "mixed clay" gradation between. Contacts between all three phases are intergradational.

Note the persistence of black biotite flakes up to the sericite front. Bright grains out in the darker colored portions of the argillized zone are light colored orthoclase crystals and direct reflections from cleavage surfaces. The veinlet (X) contains sphalerite, pyrite, quartz and rhodochrosite. ×⅓.

of the size, attitude, or comparative age of the fracture. This factual premise, interpreted in the light of definite knowledge of the relative ages of fracturing and mineralization acquired from faulted intersections, will be used for formulating the hypotheses advanced in this paper: (1) that expansion of the total alteration envelope took place by continuous reaction between the wall rock and the ore-bearing solution in the channel; (2) that there was no fundamental qualitative change in the character of the original solution, relative to its capacity for producing the observed mineralogical changes in the quartz monzonite, over the entire period of active circulation in the channel; and (3) that as long as mineralization persisted on a significant scale in the district, each alteration zone maintained its identity and relative position by growing at its advancing edge while simultaneously receding at its inner edge because of the encroachment of the minerals of the next innermost zone. The varying mineralogical identity of each zone is thus visualized as merely the result of the attainment, at the advancing front of that zone, of any combination of conditions demanded by the true stability ranges of the minerals that are forming. Heat and some chemical ingredients came from the solution; other chemical requirements were met by the wall rock itself.

Following a major persistent vein from the peripheral area of the district toward the central zone of mineralization, or following it from high elevations in the mines into depth, one quickly perceives a widening of its alteration envelope. A width of 10 ft in the peripheral area may become 100 ft or more in the zone of intermediate mineralization intensity. This change is, of course, accompanied by a change in vein content following the district's zonal pattern of metal distribution as described by the senior writer in 1913.[3]

Belts of fresh rock between parallel major structures therefore become narrower going toward Butte's Central zone until finally the outermost alteration effects from two reference veins coalesce, and fresh quartz monzonite may be found only in rounded "residual boulders" ensheathed in concentric shells of the alteration products bordering the fractures which isolated them. With continued increase in intensity conditions, one by one of the mineralogical zones of alteration is flushed out by this progressive overlap. Fresh quartz monzonite disappears first, then the green montmorillonite subzone, and finally the kaolinite subzone until only sericitized rock remains. The logical conclusion of this gradual expansion of the sericite envelope around the veins—a more or less uniform zone of high intensity alteration extending from one vein to the next—is achieved in the Central zone at Butte. This zone has made Butte famous as an example of large scale sericitization.

Obviously, the foregoing generalized picture is vastly complicated in detail by variations in closeness of spacing of fracturing. The closer the spacing the more rapid the progress of elimination of fresh rock and low grade alteration products, both because of the greater channelizing effect on the migrating ore fluid and because of the fact that the advancing alteration zones did not have to go so far between fractures.

For the reasons cited earlier in this paper it was decided to begin determinative work on the alteration products around small veinlets in places where complete suites of alteration samples, including vein and fresh rock, could be obtained. Preliminary microscopical work demonstrated the then rather surprising fact that clay minerals of several groups were important constituents of certain of the alteration zones even on the deepest mine levels. Some of these clays were of the type susceptible to base exchange and rapid changes in hydration. Small quantities of lime-rich carbonates soluble in acid mine waters were found also, and rapid postmine oxidation of sulphide alteration minerals was noted. It was then judged necessary, in order to raise reliability of the samples to the best possible level, to collect specimens as soon as the smoke was cleared from a blast in new country and seal them in paraffin on the spot. Large specimens were taken, overlapping when necessary to include all phases of all the alteration zones from veinlet to fresh wall rock. Their exact position and location were recorded on a sketch of the occurrence drawn to an appropriate scale.

## LABORATORY PROCEDURE

Three sets of laboratory preparations were made from these samples; for chemical analyses, for examination in thin section and polished thin section, and for mineralogical determinative work.

The samples for chemical analyses were sawed from the large specimens into the form of rectangular blocks weighing at least 500 g each. When possible in these studies of narrow alteration envelopes, each block was cut to include the entire width of a single mineralogical alteration zone. Otherwise, if the zone were too wide, it was cut from a megascopically homogeneous area at the center of the zone. The sawed block was then again lightly coated with paraffin and its specific gravity measured. The paraffin was used in this determination to include voids and to prevent decrepitation of the water-swelling clays upon immersion. After removal of the paraffin by sawing off thin slices of the block, the sample was crushed in a case-hardened steel diamond mortar and a 50–75 g split pulverized to minus 100 mesh by hand in a mullite mortar. This split was submitted for chemical analysis. Several hundred analyses have been made in the course of this study.

Thin sections were made for conventional purposes, including quantitative estimates by the Rosiwal method of the mineralogical compositions of the various alteration zones. In narrow alteration envelopes, where a continuous sample was available, these sections were cut to overlap one another, in order to make possible a complete study of sequence relationships across individual alteration zones and zone transitions. Polished thin sections[5] were used to study the sulphidation of ferromagnesian iron.

Mineralogical determinative work, especially on the clay mineral groups, was done on fresh material by means of standard optical procedure including double variation methods, and by thermal analyses of centrifuged gravitationally separated samples. Supporting information was derived from chemical analyses of these separations, and from X-ray powder crystal patterns.

Samples from wide alteration envelopes around large veins were treated in the same way except that they were chosen underground as representative on a basis of megascopic appearance. They were then located on a sectional view of the vein and its attending alteration zones to the scale of 1 in. equals 10 ft. The underground note sheets, with their lateral control of distance from the vein, supplied the base for graphic presentation of the quantitative chemical and mineralogical data obtained during the investigation of a suite of samples. These cross-sectional views, with specimen locations, are transferred on the same scale to a large sheet of coordinate paper (Fig 6 to 10). On this sheet, above each sample location and on the vertical line which intersects it, are plotted the specific gravity of the sample as determined before it was pulverized and the mineralogical percentages obtained from Rosiwal analyses. Below the sectional view, also on the vertical ordinate of each sample, is plotted the chemical composition of the sample, each radical against its own base line.

The value of chemical analyses to this study consists principally of gaining an estimate of the quantitative interchange of chemical constituents between ore solution and quartz monzonite at different distances from the vein. This necessitates comparison of the amount of each radical in a given zone with the amount of that radical in the original rock and with the amounts in samples from other alteration zones of the same envelope. But the specific gravities of the altered wall rocks usually differ considerably from those of fresh quartz monzonite as well as among themselves. Therefore, unless there are proportionately compensating volume changes between fresh rock and altered, gravimetric percentages as reported in chemical analyses are not directly comparable for appraisal of gains and losses. But evidence obtainable on the matter indicates that no significant volume change has taken place during alteration. There are no megascopic nor microscopic compression nor slump fractures visible in areas of widespread alteration. Quartz and orthoclase surrounding completely argillized plagioclase grains (in which specific gravity decreased from 2.65 to 2.1) show no strain nor other disruption of original rock texture. And despite intense chemical and mineralogical change, the average distance between original rock quartz grains in altered quartz monzonite shows no increase over the average for fresh rock.

Proceeding then on the common assumption of constancy of volume, comparative values of chemical content are most conveniently expressed in grams of each radical per cubic centimeter of rock. The gravimetric percentage of each radical quoted in the analyses, multiplied by the specific gravity of the reference sample in accordance with the foregoing discussion, is plotted against its own base line on the vertical ordinate drawn through the sample location. At the start of this study complete analyses were made, but after some experience was gained in evaluating the relative importance of the various radicals present, the group was cut down to the following: $SiO_2$, $Al_2O_3$, total Fe, FeO, MgO, CaO, $Na_2O$, $K_2O$, $H_2O$, $CO_2$, S.

In order to estimate the gains and losses for each alteration zone, and for the alteration envelope as a whole, the quantity of each radical in grams per cubic centimeter for fresh quartz monzonite is also drawn as a line across the suite of samples. Unless unusually good specimens of fresh rock were available from both sides of the vein, average values for composition of the fresh quartz monzonite were used for these "norm" lines. This is permissible because the wall rock is fortunately homogeneous over large areas of the camp. Evaluation of the areas between this norm line and a curve connecting the appropriate points of variation in altered rock permits the estimation of the gains and losses each zone has suffered.

## ALTERATION MINERALOGY

### *Montmorillonite Subzone*

As fresh quartz monzonite out from the wall of a fissure first came under the influence of the advancing front of hydrothermal alteration, it encountered a solution whose original chemical character had been very drastically changed by interaction with the rock closer to the fissure. At this very outer edge of an alteration envelope, the ferromagnesian minerals, hornblende and biotite, were the first to show a response to the changing environment. Marginal transition to chlorite in the biotite, and to chlorite, carbonate and tiny specks of epidote in the hornblende, are the first mineralogical changes observed. The solutions which accomplished this fringe alteration carried enough $CO_2$ to fix as carbonate part of the lime liberated from the hornblende. Except very occasionally, epidote is quantitatively subordinate to carbonate. A little iron separated out as magnetite from both biotite and hornblende during this process, and a grid of rutile needles formed in the altering biotite from $TiO_2$ displaced from the mica by base exchange. Some of the magnetite is frequently sulphidized in the fringe zone. This beginning of hydrothermal attack caused little or no megascopically visible change in appearance of the fresh quartz monzonite.

The first sign of alteration in the other rock minerals marks the beginning of the process of argillization. Tiny wisps of a montmorillonite-type clay appear along the cleavage zones of the plagioclase. This clay mineral imparts an olive-green or bluish-green tint to the quartz monzonite, the color deepening with increasing amounts of argillization. From the start, the attack on plagioclase is guided methodically by two loci of original plagioclase structure-cleavage and composition zoning. Calcic zones ($An_{40}$ to $An_{48}$) are preferentially attacked first, whether they comprise the core of progressively zoned crystals or intermediate rings in grains where oscillatory zoning has occurred. Plagioclase cleavage generally also controls the orientation of the tiny new plates of montmorillonite with the result that beautiful plaid texture is frequently exhibited by the new clay growths.

By the time argillization of the plagioclase approaches completion, hornblende has been entirely changed to carbonate and a nontronite-type clay. Alteration in biotite is still of the chlorite type, though usually it is far from complete and no longer very active. Orthoclase is unchanged.

The last residues of plagioclase in the network of montmorillonite-type alteration are frequently replaced by a series of amorphous clays having gradational properties even in small grains. The lower range of the index of refraction of this amorphous mineral is higher than that of ordinary allophane—about 1.50 in freshly broken samples—and it slowly rises to nearly 1.52 on exposure. Measurements of over 1.56 have been made on some samples of darker color. This allophane-hisingerite type mineral is often deep azure blue on freshly broken surfaces underground, but after a few hours of exposure the blue color fades to a dark olive drab or even a nearly black hue which is common in plagioclase grains in this zone of alteration. Probably both oxidation and changes in hydration influence the color change. The amorphous clay is always quantitatively subordinate to montmorillonite, even at the former mineral's peak development as a first product of residual plagioclase alteration. It decreases rapidly in relative proportion toward the vein, and its place is taken by more montmorillonite.

The montmorillonite-type mineral is the principal source of the greenish color of the so-called chloritic zone at Butte. It appears to be optically homogeneous but all of its measurable properties vary with respect to position in the montmorillonite subzone. All grains which were large enough to be tested were recognizably biaxial negative with a small 2V. The beta index of refraction is close to 1.53 for much of the material studied, though variations between 1.52 and 1.55 have been observed. Samples from near the center of the zone where this mineral is developed show the minimum index and also the strongest tendencv for water swelling. The average index rises toward the vein with decrease in birefringence and lighter green color and also toward fresh rock with darkening of the green color and higher content of iron. Varying concommitantly with increasing refraction nearer the vein, however, is specific gravity of the clay mineral. The average gravity determined is about 2.20 with extremes of 2.07 and 2.40 as determined by bromoform separations.

Thermal analyses have provided a means for coordinating most of these variable properties into systematic groups and for suggesting interpretations of mineral identity. Samples prepared for thermal analyses were pulverized as needed to get reasonable separation when centrifuged and then held at standard relative humidity (31 pct) for 24 hr. A furnace heating rate of 10°C per minute was used on all runs.

Curve No. 1 (Fig 2), of a centrifuged gravity split between 2.08 and 2.16 from an argillized rock near the center of the greenish montmorillonite subzone, compares closely with data published by Grim and Rowland[6] for montmorillonite, having a low temperature doublet, unit endothermic peaks at 700 and 900°C, and a well-defined exothermic peak at about 950°C.

The endothermic peak at 550°, however, is not a montmorillonite peak. Following through curves 2, 3, 4 and 5 (Fig 2), taken

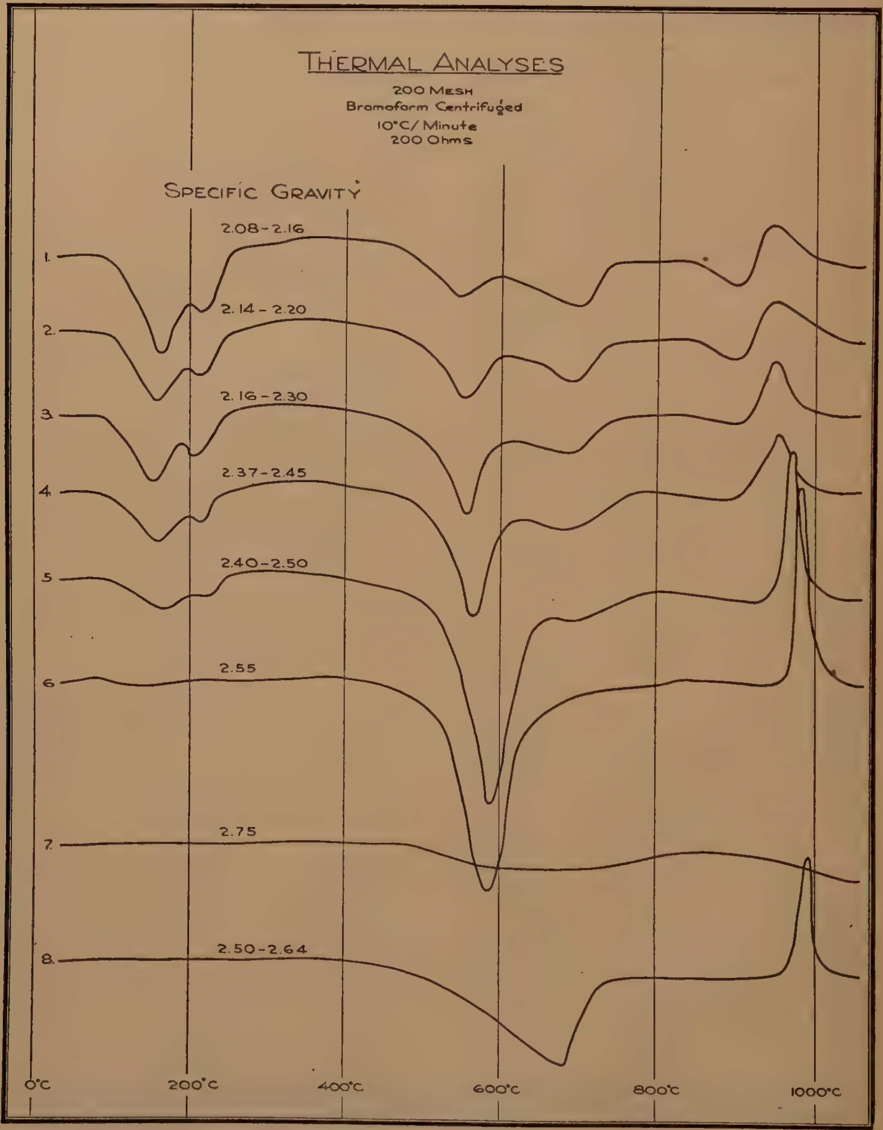

FIG 2—THERMAL ANALYSIS OF TYPICAL BUTTE ALTERATION MINERALS.

in the same argillized zone of alteration from samples successively closer to the vein, one notes a gradual deepening of that peak and a progressive fading of the montmorillonite peaks. Also, there is a considerable sharpening and displacement toward higher temperature of the final exothermic peak, until in 5 the characteristics of the curve are those of *kaolinite* with a minor contamination of montmorillonite. X-ray powder patterns permissively accord in this estimate, and, as noted on the curves, the gravity has increased through this set of samples from 2.08 to 2.44 while the color has bleached from medium olive green to nearly pure white. This set of samples is typical of many that have been studied. Yet much of the mineral appears to be optically homogeneous. Hence, a multilayered repetitive alternation of the two lattice types, kaolinite and montmorillonite, in variable proportions to give the appropriate thermal reactions, seems to the writers to be the most promising possibility of identity in evidence at present. This mechanism has been proposed by Grim and Rowland[7] and by others to explain these anomalies for many clays of this type.

The foregoing details on mineralogy of this interesting green part of the argillized zone apply directly toward establishing certain tentative hypotheses which conclude this paper. A more complete account will follow at a later date.

### *Kaolinite Subzone*

Very little of the montmorillonite-type thermal reaction shows up in curve 6. This curve is characteristic of nearly pure kaolinite. The alteration subzone characterized by this mineral is bleached white in color and speckled with black flakes of biotite, corroded pinkish or buff colored orthoclase residues, and grayish quartz. Even orthoclase sometimes shows noticeable attack at this stage, going to kaolinite more or less directly. This kaolinitic subzone, when developed, is always at the veinward edge of the argillized portion of the alteration envelope. Actually, it appears to be an end product of the gradual increase, toward the vein, of the kaolinite content of the mixed clay mineral in the argillized zone, for even the purest white kaolinite at Butte shows some small suggestion of the presence of montmorillonite by its thermal curve or low specific gravity. The zone is best developed in district-wise areas of intermediate mineralization intensity, and is sometimes nearly absent from alteration envelopes elsewhere, where a mixed clay mineral containing as much as 20 pct montmorillonite often lies in contact with the advancing sericite front.

Associated with this kaolinite in the kaolinite subzone, biotite becomes an important alteration mineral. In a foregoing comment on chloritization of rock biotite in the outer fringe of alteration, it was mentioned that the chloritization process was arrested when clay attack on plagioclase became intense. The unaltered biotite residuals are still visible as fairly large grains in the kaolinite alteration subzone, but the chlorite fringe is driven *back* to biotite. This alteration biotite is always fine-grained and frequently distributed as a halo around residuals of primary biotite (Fig 3). Fresh biotite is thus again available for attack at the sericite front.

### *Sericite Zone*

From the viewpoint of mineralogical diversity, the sericitized zone is somewhat disarmingly less complicated than the argillized zone for it is composed chiefly of only three minerals; sericite, quartz, and pyrite, in decreasing order of abundance—provided we admit certain rather wide deviations from true muscovite optics and composition within the term sericite.

White mica directly replaces all remaining silicate minerals in the argillized rock. It occurs as unit pseudomorphs after biotite, without the appearance of any chloritic intermediate phase, and also as aggregates of tiny plates, mixed with quartz, after the feldspars, whether or not the latter were first completely argillized. Microscopically, the encroachment of sericite on the clay minerals at the advancing front of the sericite zone is in evidence from textures which show sericite apparently cutting earlier clay (Fig 4).

Sericite rarely attacks the original quartz of the quartz monzonite, and in sericitized rock the total quartz content always exceeds the average amount of rock quartz. The intimate association of finely granular $SiO_2$ with sericite in feldspar areas suggests that the silica was, in part at least, coprecipitated with the sericite during the process of sericitization.

Locally, especially near quartz-rich veins in the district's Central zone, silicification becomes quite intense and silica assays of 80 pct are not uncommon in rocks still showing recognizable granitic texture. These rocks become very hard and are generally uniformly medium gray in color. Replacement vein material is the obvious end result of continued application of this silica invasion.

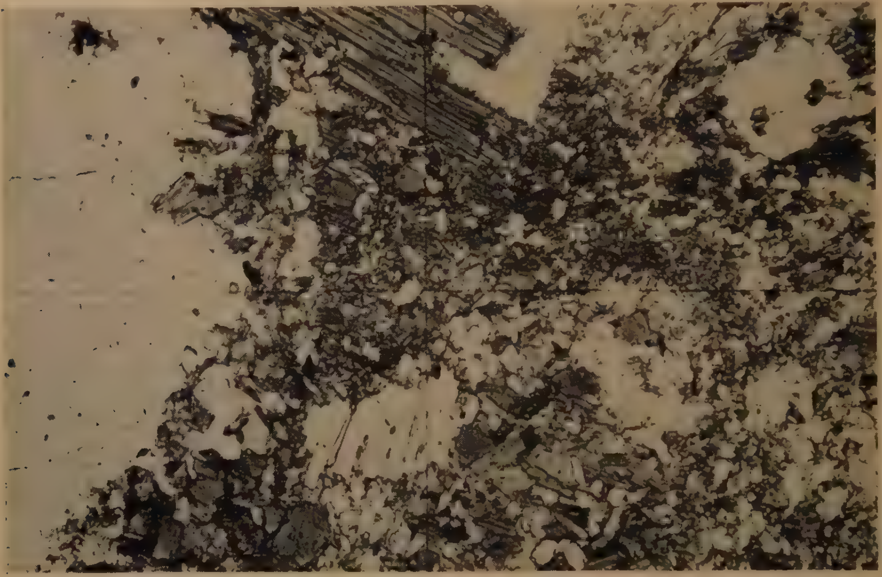

FIG 3—ALTERATION BIOTITE

Two residual grains of original rock biotite are present in this cluster of finely granular alteration biotite. Both are on the N-S cross hair, one at the top of the photograph and the other midway between the center and the bottom. Marginal and transectional corrosion are both apparent in the latter grain. Alteration biotite is the apparent replacing mineral but an intermediate stage of chloritization, here completely reconverted to biotite, probably actually accomplished the corrosion of the rock biotite. Plane polarized light. × 40.

Microscopically, this silicification quartz commonly engulfs and replaces sericite, whose alignment may still preserve original plagioclase structures two generations removed (Fig 5). This is textural evidence of the aggression of silicification on sericitization, in the same way that sericite had advanced on the clay zone at the point of reference earlier in the mineralization process.

Since it involves no change in type of mineralogy, merely a change in relative proportions between two associates because of rather local conditions, these areas of silicification are properly regarded as sub-

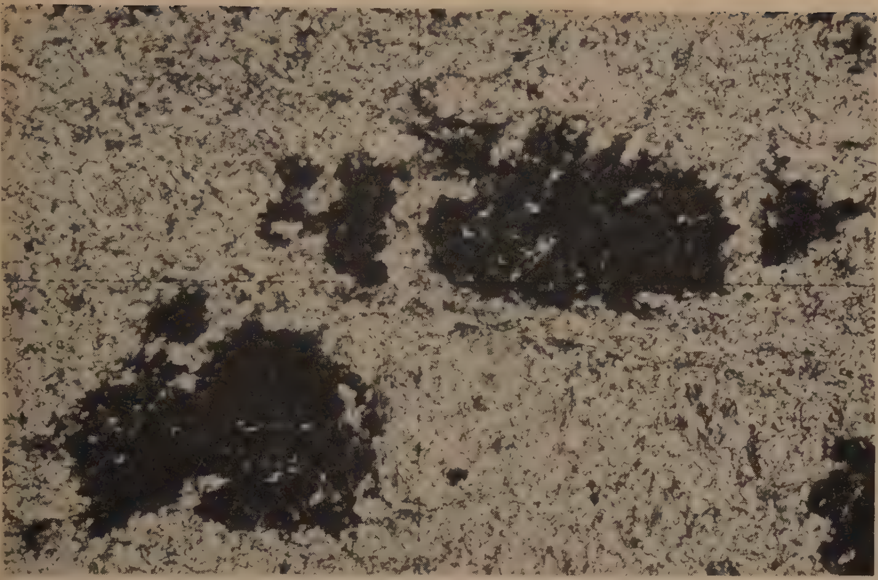

FIG 4—SERICITE APPARENTLY CUTTING KAOLINITE.

This space relationship is commonly shown in crystal areas which were once rock plagioclase in the zone of the active advance of sericitization on argillized quartz monzonite. Kaolinite appears dark and sericite light colored because of the difference in birefringence between the two minerals. Crossed nicols, × 100.

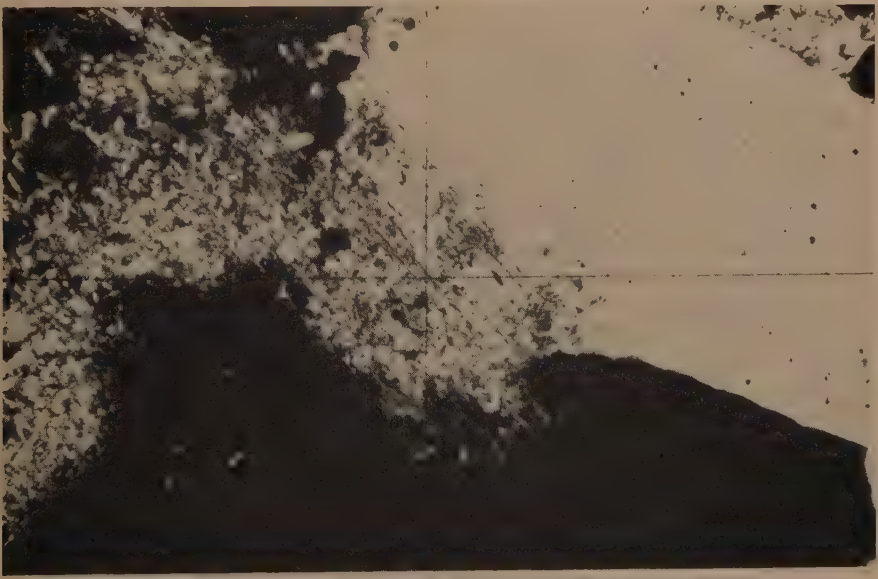

FIG 5—"GHOST" OUTLINE OF FELDSPAR EUHEDRON PRESERVED BY SERICITE.

It is noteworthy that only two quartz crystals (one at extinction position and one bright) make up the entire background of the photograph. In this manner sericite is engulfed and displaced by quartz during silicification. Crossed nicols, × 40.

zone members of the sericitized alteration zone, like the kaolinite subzone of the argillized zone.

Most of the sericite at Butte, including the greater part of that which comprises the pervasive sericitization of the district's Central zone, shows a fairly good approximation of the properties of muscovite, except that it is somewhat low in alkali content, in indices of refraction, and in specific gravity. Its thermal curve (Fig 2, curve 7) is characterized, in general, by a broad, shallow trough centering at about 600 to 700°C, and a sharper endothermic peak at about 1040°C. The lower the alkali content, the narrower and sharper is the lower temperature endothermic peak, though the water content of the mineral does not go up more than a few tenths of a per cent. A pure sample of this latter mineral from a vug filling on a deep level in the Central zone shows a total of 8 pct alkali radical, including 1 pct $Na_2O$. The lowest index of refraction falls just below the higher index of quartz. This high-intensity low-alkali sericite is the only case of the fixation of $Na_2O$ other than in montmorillonite observed to date in Butte alteration minerals. Low-alkali sericite is also an important constituent of the altered walls in deeper mine levels.

A common associate of this type of sericite, both as a vein constituent and in the wall rocks near the hot center of camp, is the high-intensity clay mineral dickite, identified optically, by means of its characteristic thermal curve (Fig 2, curve 8), and by X-ray powder pattern. Good pink crystalline alunite is also sometimes found in this association, though little of it penetrates very far into the wall rocks.

Both dickite and low-alkali sericite are commonly interlaminated with beautiful covellite plates and rich crystalline masses of digenite and chalcocite in the deep levels of exploration in the eastern part of the district.

## CHEMICAL AND MINERALOGICAL CHANGES IN THE WALL ROCKS

As stated earlier in this paper, our attempt to evaluate the chemical interchange and mineralogical relationships between the ore fluid and the walls of the Butte veins has made use of detailed cross sectional diagrams like those of Fig 6 to 10. Several hundred such sections across minute veinlets, major structures, and all gradations between, have been studied. In general, as we have already pointed out, they all tell the same story, and the five sections from ore-bearing veins selected to accompany this discussion are representative of the types of activity that are characteristic of the Butte district.

The cross section of Fig 6 is from a 1900 level crosscut on a prominent East-West vein in a locality exhibiting near-peripheral type of mineralization intensity. The metallization is zinc ore composed of sphalerite, galena, pyrite, and quartz. The section of Fig 7 is from the same vein farther west geographically but from a much deeper level, the 3800, where the metallization is copper ore with minor zinc. In each case the alteration envelope includes zones of sericite, kaolinite, and montmorillonite, named in order of their position measured outward from the vein and generalized in the cross sections. Comparing the sections, it may be observed that in spite of the sharp differences in metallization of the vein at these widely separated localities, the chemical action on the wall rock and the mineralogical results effected by alteration processes working outward from the vein are qualitatively identical, although the total width of the alteration envelope at the 3800 level is about three times that at the 1900 level.

In both sections the lime, soda, and silica curves show strong decline toward the vein across the argillized zone. The loss of magnesia by the walls is in the same direction but less rapid. Mineralogically, the

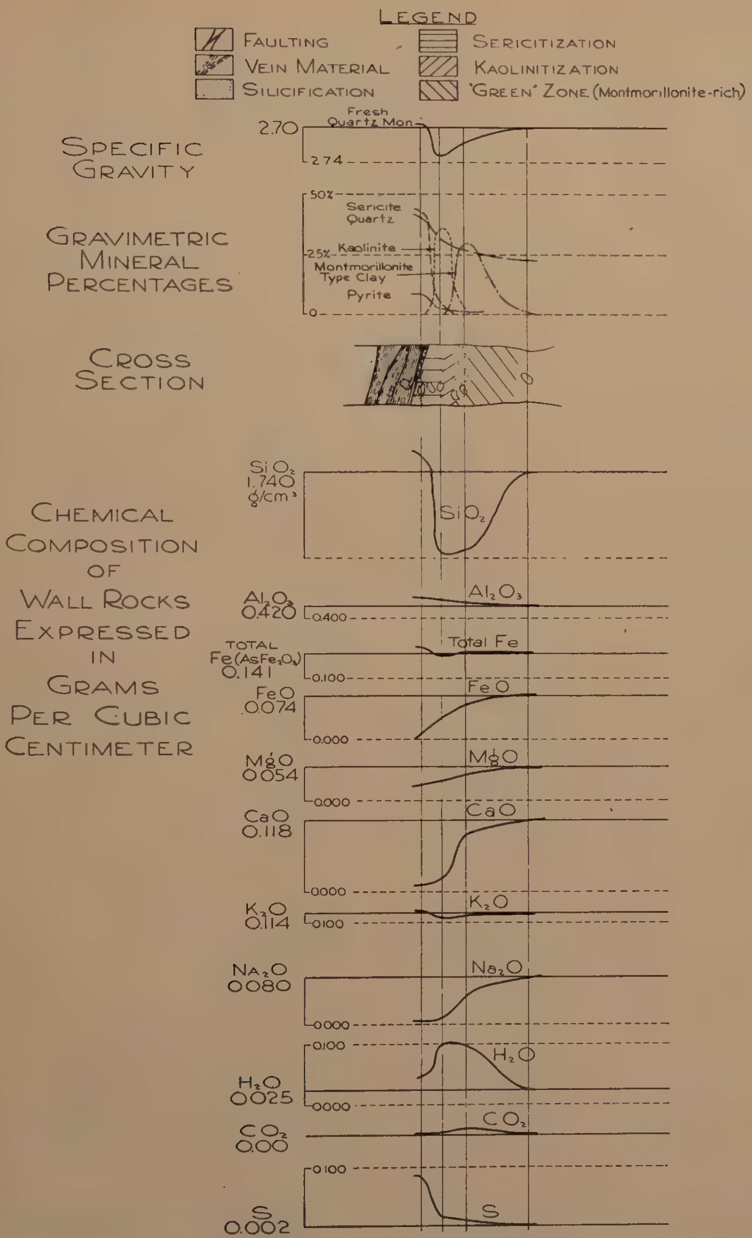

Fig 6—Cross section of an East-West vein, 1900 level.

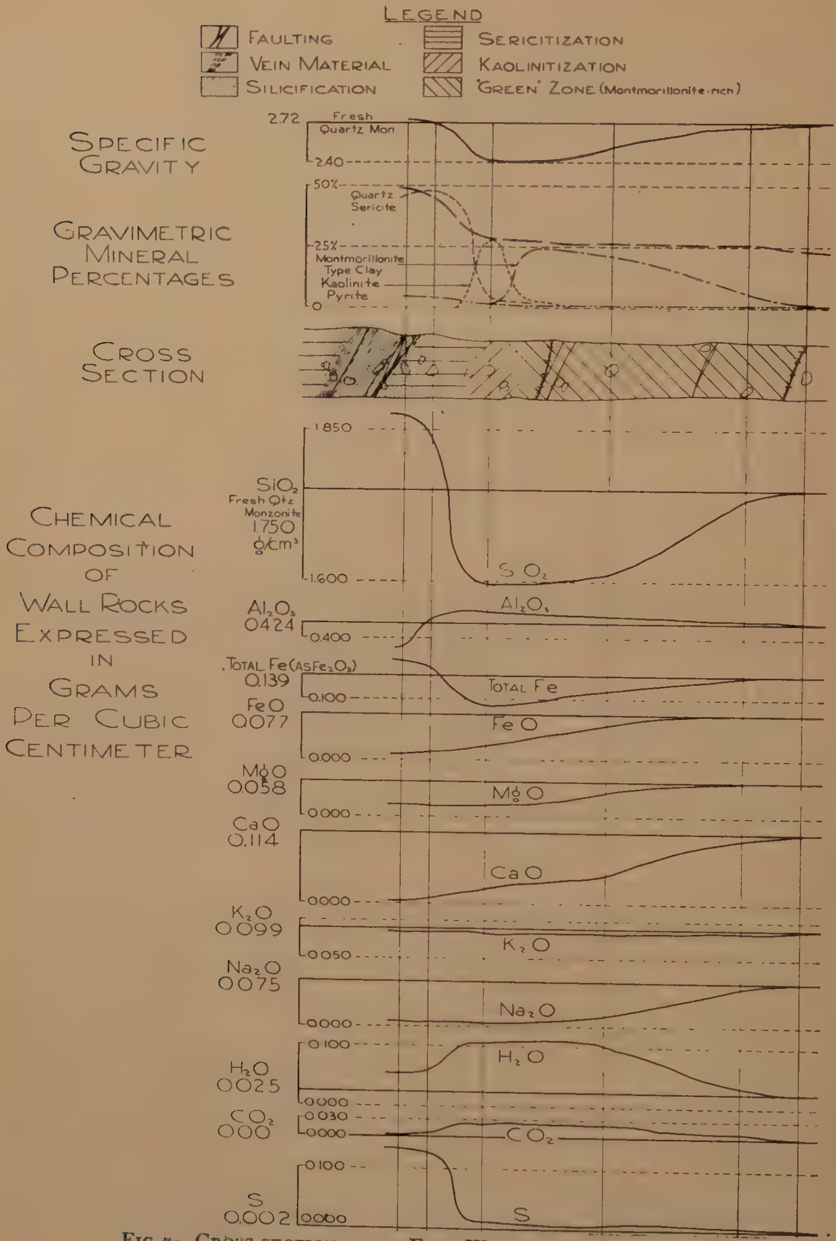

FIG 7—CROSS SECTION OF AN EAST-WEST VEIN, 3800 LEVEL.

losses of the above-mentioned radicals are evidenced by the progressive destruction of plagioclase with the temporary fixation of part of its constituents plus some magnesia and iron in the mixed clay mineral. The increasing proportion of kaolinite as compared to montmorillonite in the mixed clay, proceeding toward the vein through the argillized zone, is demanded by the continuing abstraction of the foregoing radicals during the gradual outward expansion of a progressively stronger leaching environment.

$K_2O$ and $Al_2O_3$ remain significantly close to the lines defining the composition of fresh rock. $K_2O$ is held up principally by the metastability of orthoclase in this environment, since the latter mineral shows only partial breakdown even in the kaolinite subzone. There is also a little potash feldspar in solid solution with the plagioclase and the potash from this source probably accounts for much of the decline that is shown by the $K_2O$ curve. The proportion of alumina increases in the progressively changing clay minerals toward the vein, so, even though the specific gravity of the rock decreases, the alumina curve stays high.

The principal gains of constituents in the argillized zone are registered in water for hydration and in $CO_2$ for carbonates, plus a little sulphur for pyrite. Hence from the point of view of the destruction of rock minerals the significant process in the argillized zone is the increasingly intense selective removal of the base and alkali metals, plus their molecular equivalents in $SiO_2$, from plagioclase and hornblende.

Except for the lowest grade start toward chloritization it received at the outer edge of the alteration envelope, a member of the biotite group is within its range of true stability and the potash feldspar is metastable in the environment of the argillized zone.

The foregoing subtractive process operating in the argillized zone requires the presence of constituents in the fluid medium of transfer that could break down the silicate molecules of lime and soda and, to a lesser extent, magnesia, hold these constituents in solution, and avail their transport out of the zone. The increasing efficacy of this process toward the vein indicates increasing ability of the solution nearer the vein in the argillized zone to accomplish this removal and, at the same time, to press the abstraction process toward completion. The result over a period of time was a gradual pushing outward of the front of attack as the total envelope of alteration grew wider while maintaining the identity and positions relative to each other of its component zones.

Continuing supply of the ingredients required to push the alteration zones outward must have come from the channel, and spent products of reaction were removed to the channel. This two-way transfer could only have been accomplished by ionic diffusion. Because of the evident continuing reaction and solution of base and alkali silicates, illustrated by the gradual but persistent decline of the base and $SiO_2$ curves, and resulting in the gradual displacement of montmorillonite by kaolinite in the mixed clay mineral of this zone, there would be a tendency toward chemical neutralization of the solution progressively farther from the vein. Fulfillment of this tendency might be indicated by the low-intensity epidote and chlorite in the outermost alteration fringe.

Pertaining to the hypothesis mentioned earlier in this paper that sericitization and argillization are contemporaneous processes, Fig 9 illustrates a cross section from a Northwest vein which *offsets sulphide mineralization* on the East-West veins of Fig 6 and 7. The fundamental similarity of this diagram to those from the East-West vein is obvious both as to quantitative chemical variations and distribution and identity of mineralogical products. Argillization must have been accomplished on the

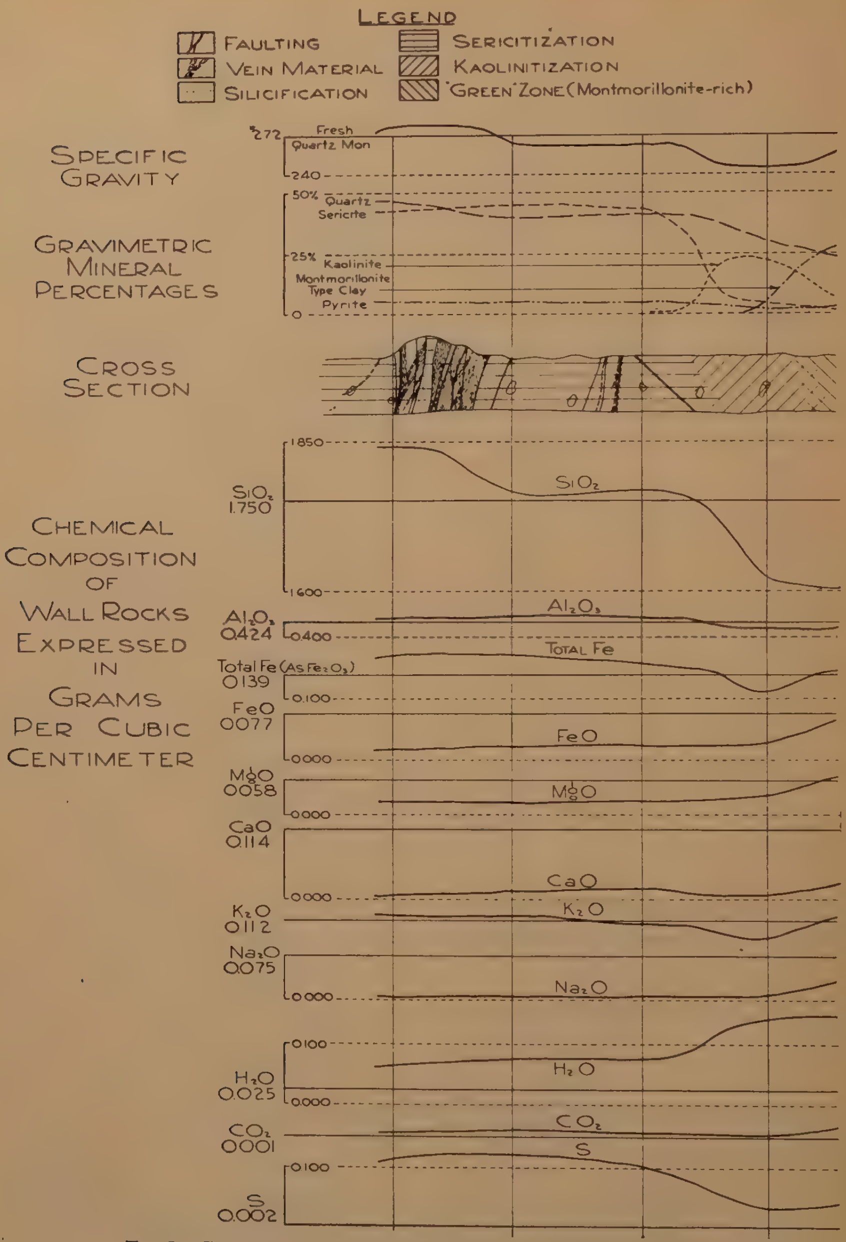

FIG 8—CROSS SECTION OF THE EAST-WEST VEIN, 4200 LEVEL.

Northwester after sulphide deposition had begun in the East-West vein which it offset. The ore-bearing solution, which then had free access to both structures, continued its work of deposition in the East-West fissure and began this process in the Northwest fault, contemporaneously attacking the walls of the latter according to the same chemical and mineralogical pattern as it had previously applied and was still applying to the walls of the earlier formed and partially metallized East-West fissure.

In every case the curves of base leaching from the walls by the solution continue their downward slope toward the vein *into* and *through* the zone of sericitization. The arguments of base-metal leaching applied to the argillized zone may be reapplied here, since the little remaining Na, Ca, and part of the Mg were still incapable of producing stable compounds in the environment of this zone. The soda and lime curves, in particular, drop to nearly zero. Potash and alumina came from the orthoclase and clay to make sericite, and some of the veinward migrating silica from the argillized zone was precipitated as quartz intermingled with the sericite, raising the silica percentage in the sericitized zone above the quantity present in fresh rock. Any excess of silica abstracted from the argillized zone was, of course, available for precipitation in the channel as vein quartz. Some quartz veins may grow at their outer surfaces by accretion of this veinward migrating silica.

The replacement of biotite by sericite is in the nature of a base-exchange phenomenon, for various stages of the change may be observed in some thin sections, and unit pseudomorphs of sericite after biotite are prevalent. Sulphidation of iron liberated from the biotite also takes place immediately and contemporaneously with the sericite-forming reaction, illustrating the close association and interdependency of sulphide formation with potash-mica crystallization.

## SUGGESTED CONTROL MECHANISM FOR THE SERICITE ZONE FRONT

The big question now becomes: is the hypothesis of simultaneous sericitization and argillization at different distances from the vein compatible with what we actually know about the requirements for crystallization of the various minerals involved, particularly for sericite? Presenting permissive evidence from syntheses, it may be stated that several experimenters have formed sericite in basic or neutral solutions at temperatures as low as 200°C, and J. W. Gruner[8] has altered microcline and albite to sericite in an environment made acid with 0.1 N to 0.35 N HCl, though at temperatures of 350 to 525°C, and in the presence of excess KCl. Gruner also found that "as the acidity decreases the field of formation of muscovite probably extends to lower temperatures, but does not reach 300°C at 0.05 N." It is apparent from these two sets of data that the kindling temperature for sericite formation, assuming availability of appropriate constituents, takes an abrupt rise across the transition in pH from neutrality to slight acidity. But a point of significance is that if the temperature is high enough, that is, of the order of 350°C, sericite may form even in a decidedly acid solution (pH 1 at close of most of Gruner's experiments) where potassium is available. Below this kindling temperature only kaolinite was formed even in the presence of excess KCl.

In a neutral or slightly basic chemical environment, the position of the sericite alteration front might be determined by the passage of orthoclase from its state of metastability in the clay-forming zone into a range of chemical conditions where it could no longer maintain its identity. Its breakdown because of increasing capability of feldspar decomposition by the solution could then supply the potash for sericite. Except in a strictly permissive role, temperature (200°C requirement) may not be an important factor.

In an acid environment, however, *if* such be considered a requisite for the accomplishment of kaolinitization, two requirements especially pertinent in this instance must be met for the crystallization of sericite and for the consequent advance of the sericite front: availability of potash, and attainment of the required temperature. Much of the potash for sericite evidently comes from the orthoclase which has persisted through the zone of argillization up to the sericite front. As above, chemical attack by the sulphur-bearing ore fluid on the orthoclase, with consequent release of potash, may provide the critical control of sericite formation, assuming a sufficiently high temperature. Or, the temperature requirement itself may be the critical control, and the breakdown of orthoclase may be consequent from the vigorous crystallization of and attack by sericite when the kindling temperature of the latter is achieved. In this case, the position of the sericite front at any given time and its velocity of outward migration over a period of time may have been determined by the rate of supply of heat from the solution. Moreover, the rate of supply of chemical ingredients to accomplish the base leaching of the argillized zone, compared to rate of supply of heat, could influence the comparative widths of the sericitized and argillized zones of alteration. A relatively faster rate of heat supply would cause the front of sericitization to advance more rapidly on the argillized zone than the latter could progress against fresh wall rock, and a comparatively narrower argillized zone would result. Sericite might encroach on the clay mineralogy rapidly enough to prevent thorough base leaching from montmorillonite and the formation of a well-defined kaolinite subzone. This might be expected, for example, in areas of closely-spaced heavy fracturing where relatively rapid heating of the walls would be probable. In several such localities, particularly in and near the Central zone of the Butte district, this occurrence is well illustrated by the replacement by sericite of a clay mineral still relatively rich in the montmorillonite molecule.

Contrarily, broad areas of argillization cut by narrow bands of sericite around intersecting veinlets are common in the west central intermediate zone of the Butte district. In general, such areas resemble certain phases of some of the porphyry copper type of mineralization and attending alteration. The quantitative preponderance of clay over sericite at these places in Butte might have been accomplished, for example, by a chemically potent hydrothermal solution whose heat supply was relatively low; not necessarily by post-argillization veining by sericite deposited from later, radically different chemical solutions. The apparent age relationships, both field and microscopical, would be the same under each hypothesis. Merely the space-time concepts applied to their interpretation would be different. For Butte, structural proof is available for the district-wise contemporaneity of sericitization and argillization.

Adequacy of the heat supply in the hydrothermal solution at Butte for the operation of the temperature-front mechanism for sericitization is indicated by vein quartz crystallization temperatures as determined from observed temperatures of disappearance of the vapor phase in liquid inclusions. Applying Ingerson's[9] corrections for depth of burial of about 3 km, the observed temperatures range from 300 to about 380°C, bridging the temperature span of sericite formation in acid solution as determined by Gruner.

In the cross sections thus far illustrated by chemical diagrams, the losses by the argillized wall rock in silica, for example, are more than adequate to account for the silica gains of the sericitized zone of the same envelope. In the process of constant growth of the alteration zones visualized in the foregoing discussion, adjacent envelopes will eventually meet at their outer edges and overlap, with the complete eradication of fresh quartz monzonite. Meanwhile, sericitization continues to expand from the vein outward, and ultimately a position of the sericitized zone is reached where losses ascribable to a given argillized zone are balanced by gains in the widening sericitized zone at deeper levels. This condition is approximated in the cross section of Fig 8 which includes a section of wall rock measured outward from a large East-West vein on the 4200 level of the Mt. Con mine, at a point nearer the Central zone and deeper in the mines than the location of Fig 7. Here the vein carries copper but no zinc. No fresh quartz monzonite is present in the crosscut penetrating the walls of this vein because of the overlap of the alteration envelopes developed outward from the vein and from the next parallel structure 100 ft farther north. The lowest grade alteration present here is a light greenish montmorillonite-type clay intergrown with considerable kaolinite. However, a strong white bleached subzone containing nearly pure kaolinite is also present between the green clay and the sericitized zone.

Moving deeper into the district's Central zone along any of the major vein systems, chemical gains in certain radicals in the zone of sericitization begin to outweigh losses of those radicals in the argillized zone. This situation reaches its climax in the Central zone of mineralization where pervasive sericitization, increasingly wide at greater depth, plus or minus local silicification, occupies the entire distance from one vein or mineralized zone to the next. The cross section of Fig 10 from the Leonard vein deep in the Central altered zone of the Butte district illustrates the final sericitization phase of the alteration process. Here silica, alumina, potash, and iron, in addition to much sulphur, have been added to and fixed within the walls so that the total amount of each radical now present exceeds the amount present in the originally fresh quartz monzonite. Moreover, the specific gravity of the rock is higher than it was originally, so quantitatively the total gains outweigh the total losses in other constituents. These materials must have been supplied to the walls from below by the hydrothermal fluid. How the fluid acquired these constituents is conjectural, since they could either be supplied from a deeper alteration zone through another change in mineralogical equilibrium conditions or they may have been integral to the ore fluid at the place of its accumulation. This matter need only concern us to the extent of observing that the latter method is not a requirement, and that a suggestion of depletion of potash supply *may* be indicated by the low-alkali sericite and dickite in the near-vein alteration in this cross section. These minerals show up in the chemical curves as a decline in the $K_2O$ curve and a corresponding increase in the quantity of $Na_2O$, which radical, as pointed out earlier, attempts to fill part of the place occupied by potash in the mica molecule when the potash available for mica is low.

## CONCLUSIONS

In the foregoing discussion, continuity and uniformity over the period of active circulation of the ore fluid have been the emphasized attributes of the alteration process as judged from the chemical action and mineralogical results of attack by the ore fluid on its channel walls. And the suggestion has been made that the mineralogical stability requirement of the hypotheses presented, as deduced from the evidence offered by mineral syntheses, are within the known ranges of fields of formation of the alteration minerals recognized at Butte. As a concluding argument we desire to emphasize once more the structural field evidence at Butte which, we believe, requires the operation of the hypothesis of the district-wise contem-

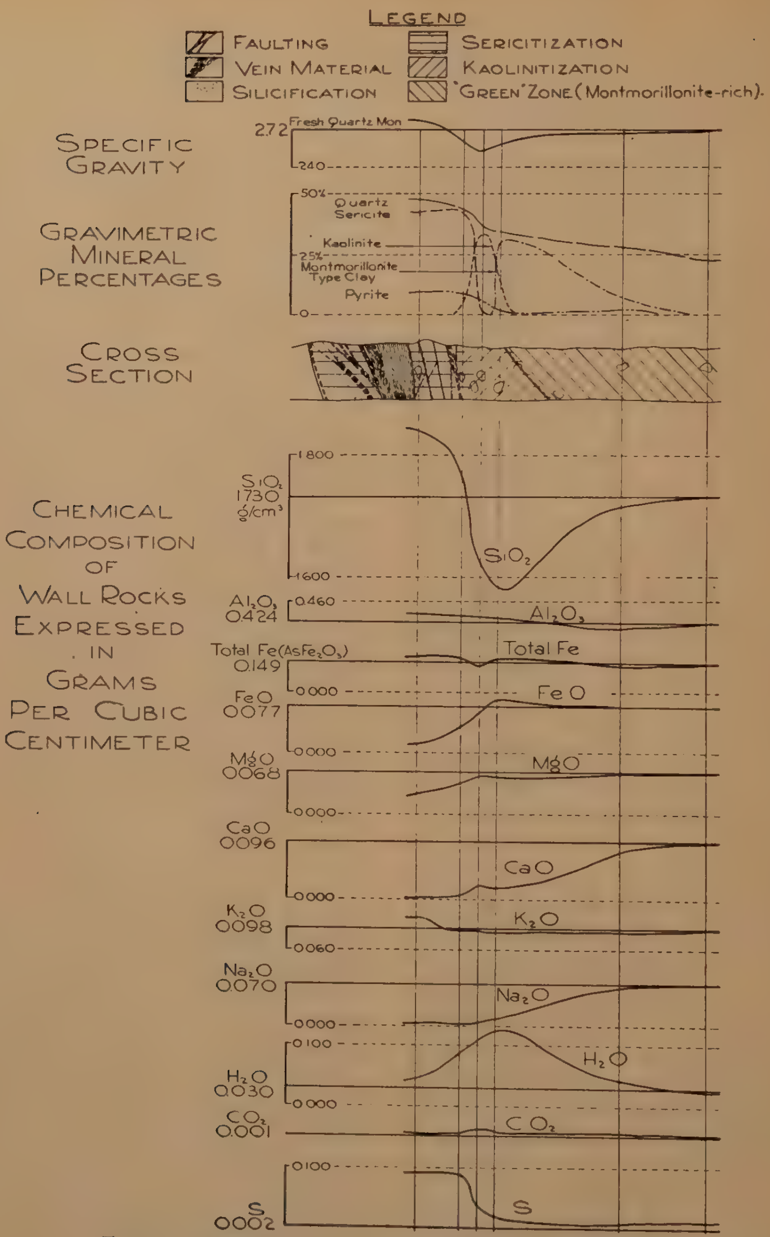

FIG 9—CROSS SECTION OF NORTHWEST VEIN, 2800 LEVEL.

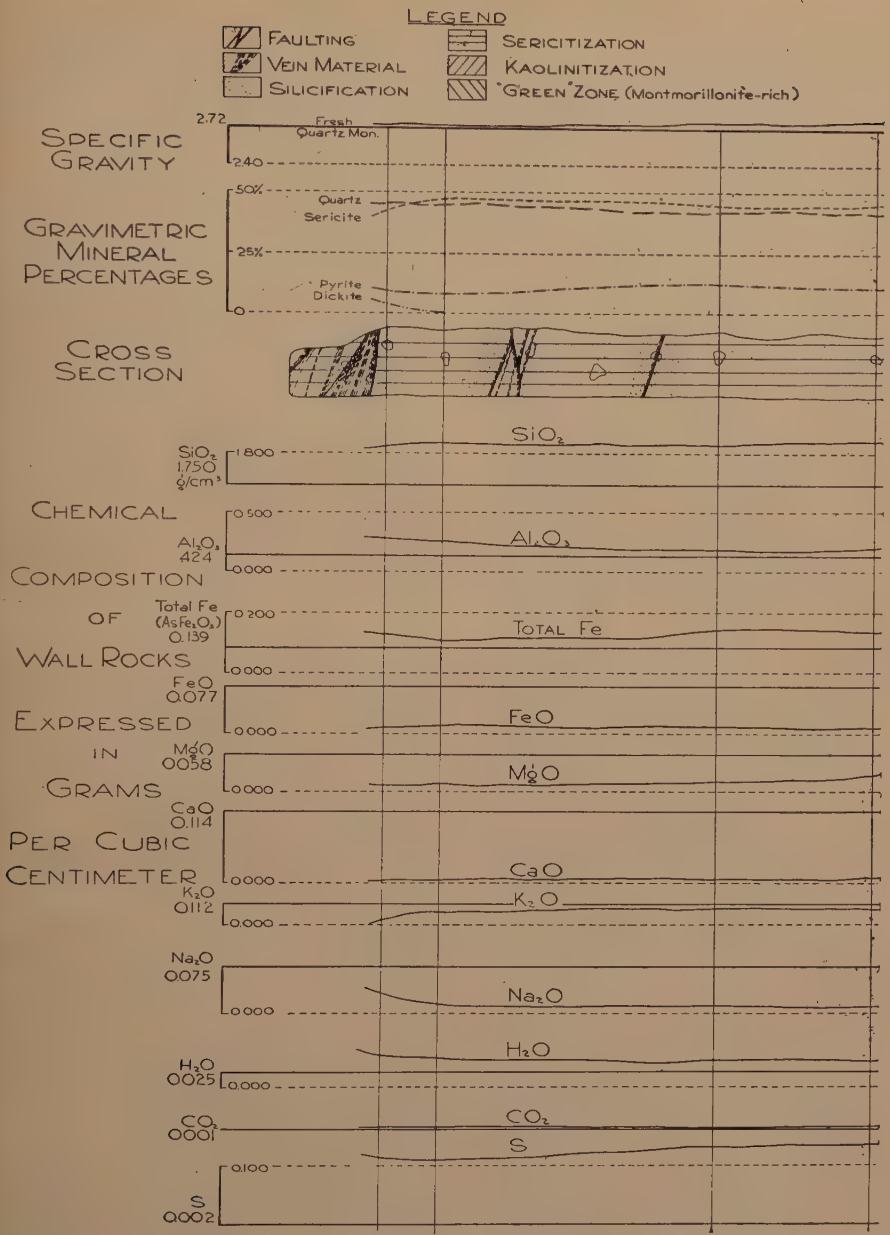

FIG 10—CROSS SECTION OF EAST-WEST VEIN, 3800 LEVEL.

poraneous formation of zones of sericitization and argillization.

It is obvious that any changes in fundamental character of the ore fluid, over the period of active circulation of that fluid through the succession of channels provided by intermittent faulting at Butte, should be recorded by the variable characters of the veins found in these faults and of the wall rock alteration adjacent to them. If an early period of alteration, characterized only by argillization, had been succeeded by one in which only sericite was formed in the walls by reason of a radical change in the composition of the ore fluid, that change should be recorded in the superimposition of sericitization on argillized rocks around early-formed fractures and by the total absence of clay minerals from the alteration zones around fractures which formed late in the period of metallization.

But, as we have stated, the observed facts of distribution of alteration products at Butte stand in sharp contradiction to this requirement. Successive zones of sericitized and argillized quartz monzonite occur around every ore-bearing fracture, regardless of size, attitude, or relative age; provided the lower intensity alteration zones are not flushed out by coalescence of the alteration envelopes of adjacent structures, as is the case in the Central zone. Furthermore, the two types of alteration always occupy the same relative positions: sericite adjacent to the ore-bearing vein; clay minerals always between sericitized rock and fresh quartz monzonite. And among themselves, the clay minerals always are arranged with the high kaolinite end of the series nearer the vein and the montmorillonite end toward fresh rock.

According to any theory based on the premise that an epoch of sericitization followed an epoch of argillization, the geometric pattern of alteration which envelopes every mineralized fissure in the Butte district would require that the invading zone of sericitization never advanced completely through the argillized zone to attack fresh rock directly. It would have had to advance only far enough out from every part of the respective channel walls to leave unchanged the characteristic band of argillization between the advancing sericite zone and unaltered rock. Furthermore, if the solutions which produced sericite were not capable of causing argillization, and if sericitization therefore wholly followed all argillization, sericitization could not have commenced until the very end of circulation in the channel maze after all mineralized fissures of all ages had been formed and sulphide deposition had all but ceased completely; for in areas where a complete structural sequence may be observed, argillization is always present, and in its customary zonal position, in the wall rocks of Northwest-faulting and Northeast-faulting fissures and even in the last formed fractures of the district along which there was any mineralizing activity. This would place sericite in the untenable position of postsulphide age and run counter to the generally observed ubiquitous close association of sulphides and sericite at Butte and in other deposits of its type.

Structural evidence also indicates that many early East-West veins were completely "filled" with quartz and sulphides before they were offset along Northwest faults. Some of these early "sealed" veins should have escaped circulation of the postulated postfracturing epoch of sericitization and be contained, therefore, in an envelope of only argillized quartz monzonite without sericite. But, instead, the facts of occurrence are that the wall rocks around all early veins are abundantly sericitized. In fact, circulation along the East-West fracture system is chiefly responsible for the Central zone of intense sericitization which has widened rapidly with depth of exposure in the mines.

Logically, as well as geologically, only one alternative explanation of the ubiquity of argillization and sericitization around all ore-bearing fractures at Butte is available. In the opinion expressed here it fits the facts of distribution of alteration products better than an explanation involving two or more distinctly different chemical solutions operating at different times with different mineralogical results. This alternative proposition demands that sericitization and argillization are contemporaneous processes; that the frontal attacks of each zone on the next outermost alteration zone were made at the same time. As long as active circulation took place in the channel, each zone migrated away from the fissure, that is, it grew at its outer edge and simultaneously receded at its inner edge because of encroachment by the next innermost zone. And the different mineralogical and chemical responses to this attack are dependent not on a drastic about-face change of composition of the fluid in the channel but on smoothly varying conditions of physicochemical environment outwardly from the vein, demarcated into mineralogical zones by responses dictated by finite mineralogical true stability ranges.

This is the simplest geometric interpretation and the only one which fits all details of distribution, apparent age relationships, and true stability ranges of the minerals themselves. It became attractive at the beginning of this study as an explanation for distribution around small isolated veinlets cutting fresh quartz monzonite, where complete point to point observation of all mineralogical and chemical changes was facilitated. It was then applied on what might be termed an intermediate scale to large veins, where complications of intersection and bifurcation of fractures are prominent. And finally, viewing the district as a whole, it is clear that just as the sericite and argillite fronts were migrating outward from a vein during the period of active mineralization, so were they migrating district-wise radially upward and outward from a deeper centralized source. The pattern, when thus generalized, is the same for all three scales of application. The higher intensity alteration zone, where minerals of the sericite group predominate, is in each scale of application closer to the source; that is, to the supply of heat and necessary chemical ingredients, where rapid reaction to form sericite is more easily achieved.

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## APPENDIX A

| Degrees | |
|---|---|
| Centigrade | Fahrenheit |
| 10 | 50 |
| 200 | 392 |
| 300 | 572 |
| 350 | 662 |
| 380 | 716 |
| 525 | 977 |
| 550 | 1022 |
| 600 | 1112 |
| 700 | 1292 |
| 900 | 1652 |
| 950 | 1742 |
| 1040 | 1904 |

## DISCUSSION

(*C. D. Hulin and P. J. Shenon presiding*)

C. D. HULIN*—It is rare indeed that one has the privilege of hearing the presentation of the results of such a detailed piece of research investigation. The authors are certainly to be congratulated, and the presentation and publication of this paper unquestionably marks a tremendous step forward in our understanding and knowledge of wall rock alteration.

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* University of California, Berkeley, Calif.

W. D. Johnston, Jr.*—This paper did not come from a university; it did not come from a governmental institution; it came from a mining company.

Mining geologists have oftentimes heard the query, "When will geology be applied in mining with the intensity and success that is used in the petroleum industry?" This paper appears to me to bear on that question, for it is an investigation of a basic geological problem, by a mining company. That it was fathered by Reno Sales, a past recipient of the Penrose medal of the Society of Economic Geologists for his own pioneering in the structural control of the ore deposits of Butte, is the least surprising thing about it. The authors, the Anaconda Copper Mining Co., and the mining industry all appear to be candidates for congratulations.

L. C. Graton†—Many of us, I am sure, agree with the Chairman that this is, indeed, a rare occasion. As for Dr. Johnston's comments: I believe a little reflection will make it clear that so extensive and penetrating a research as this could have been performed only right on the job and with the abundant resources available to a mining company.

As this most impressive story developed, I kept feeling that there has been nothing to approach it as exploration into the nature and significance of rock alteration since Lindgren's classic "Metasomatic Processes in Fissure Veins,"[10] presented before the Institute almost 50 years ago. Here is a most imposing amplification of Lindgren's approach to that important problem, and a fine vindication of his fundamental philosophy that great light can be shed on the conditions of ore deposition by study of the alterations produced in the adjacent wall rock.

It is significant that before Anaconda—which, geologically speaking, has for half a century been synonymous with Mr. Sales—was willing to undertake this kind of geological study at Butte—I will even go so far as to guess that before Mr. Sales had much use for such minerochemical study at Butte—he insisted on getting his fundamental structure and his structural timing pretty nearly right. The very beautiful argument that Mr. Meyer has presented with respect to the timing of the successive phases of rock alteration, gains enormously, as I see it, in its impressiveness and its degree of approximation to the truth by its striking compatibility with, and confirmation of, the structural complexity and time difference that everyone now accepts at Butte. On this account, Butte was a unique and highly significant place, quite apart from the mineralization itself, for this kind of study.

During the latter part of the war and since, we have heard a good deal from scientists who emphasize the inherent contrast between fundamental research and applied research; and there has been a widespread wail that while such a vast amount of research was conducted during the war, it was essentially all applied research, and basic, fundamental research languished.

While listening to the paper just presented, I have been trying to appraise what kind of research it represents. By the sharp distinctions which some of these scientists tend to draw, I suppose that establishment of the concept of differential thermal-analysis and the development of the equipment for its application would be accepted as a piece of fundamental research. But to utilize this and analogous conceptions, instruments and procedures already established, as a step toward elucidating the origin of the Butte ores, would probably be placed by some in the category of applied, commercialized research. I would be inclined to challenge such a distinction as improperly literal and narrow. It seems to me that the report we have heard this afternoon is as fresh, as inherently scientific in the true sense, and as rich in its possibilities for further illumination of basic geological problems, not only at Butte but at countless other places, as almost anything we can imagine. If this is not fundamental research, then we may well inquire not only what better examples may be cited in the domain of geology, but also whether there is a place for fundamental research in geology.

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† Harvard University, Cambridge, Mass.
[10] W. Lindgren: Metasomatic Processes in Fissure Veins. *Trans.* AIME (1900) **30**, 578.

W. D. Johnston, Jr.—May I take this opportunity to voice a plea for more temperature data on mineral deposits. A recent paper by Earl Ingerson[11] describes a method for estimating temperatures from liquid inclusions in minerals. We need several thousand such temperature observations, both to check the validity of Ingerson's method and to give a more substantial basis for our badly worn ideas about temperature classification of ore deposits. This plea is directed to the professors here. Will you not see that your graduate students are on the lookout for microscopic liquid inclusions in whatever material they are working on and that they record whatever temperature data they so obtain? If it is not possible to make temperature determinations in your institution, Dr. Ingerson, at the Survey, would be happy to do so.

Perhaps you will bear with me for an additional remark about wall rock alteration. Recently I saw some magnificent color photographs taken from the air by the Aero Service Corp. Color photography appears to offer a useful tool for the precise mapping of alteration zones at the surface. There is need for someone to try it out.

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[11] Earl Ingerson: Liquid Inclusions in Geologic Thermometry. *Amer. Mineralogist* (1947).

# Factors in the Localization of Mineralized Districts

By Carlton D. Hulin,* Member A.I.M.E.

(New York Meeting, February 1944)

## Abstract

The usual concurrence in time and space of intrusive igneous activity, favorable structural disturbance and mineralization, which is so manifest in the mineralized district necessarily indicates a close interrelationship between these events. Evaluation of available evidence leads to the recognition that an appreciable time interval exists between the emplacement of the igneous intrusive and the formation of associated ore deposits. During this interval known portions of the intrusive are completely solidified, highly differentiated magma facies are injected as dikes, and important faulting occurs, which may fracture and shatter the solidified intrusive body.

The highly differentiated dike types could only have originated in some deep portion of an underlying plutonic body that had reached an advanced stage of magmatic differentiation. Appearing after these dikes, the mineralizing solutions may likewise be presumed to have originated at great depth. Certainly no source for these solutions can be found in near-by portions of the intrusive igneous body.

The association of igneous intrusives and clustering ore deposits must be recognized as resulting from structural control. The emplacement and the subsequent solidification and cooling of the intrusive set in operation forces that are responsible for the favorable structural disturbances that control the mineralized district.

The emplacement of the body of fluid magma weakens the crustal rocks to the extent that failure due to strains originally present in the rock body may occur. Faulting of any type and of any magnitude may be the result. Subsequently, owing to the change of state from fluid magma to crystalline granitic rock, volume decreases approximating 10 per cent may occur, with resultant shrinkage and collapse within the rock body.

Following solidification, thermal contraction, induced by cooling of the rock body, causes a further volume reduction of 6 per cent. This thermal contraction, extending over a lengthy period of time, results in a succession of minor fault movements. These small fault movements, occurring progressively throughout the period of mineralization, are responsible for maintaining the permeability of the rock body, without which access of the mineralizing solutions and the formation of the ore deposits of the mineralized district would be impossible.

## Introduction

A proper understanding of the causes that have localized our mineralized districts should be of considerable value to geologist and engineer alike. Under current conditions the known ore reserves of productive mining camps are being rapidly depleted. Although ore of the future will be derived in part from deposits now known and from ore bodies that are still to be discovered within the limits of productive mining districts, in part at least it must come from mineralized districts whose existence or importance yet remain to be recognized.

The mining geologist is thus faced with two problems of paramount importance: (1) the location of new ore bodies within known mineralized areas and (2) the location of new mineralized districts within whose confines, previously unknown, valuable ore deposits may occur. Any contribution to our knowledge concerning the causes for the localization of mineralized districts

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Manuscript received at the office of the Institute Feb. 5, 1944. Issued as TP 1762 in Mining Technology January 1945.

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should have an important bearing upon such discovery.

While the two problems differ in many important respects, it must be recognized that they possess one fundamental feature in common: both are four dimensional, involving as they do three-dimensional space structures present in the rock body, and the continuously changing and varied sequence of events that have affected these structures with passing time.

## THE PROCESS OF MINERALIZATION

Geologists of today are largely in agreement that mineralization is genetically related to igneous activity. It is held that, with few exceptions, the mineralizing solutions, highly dilute and largely aqueous in nature, originate at depth in some plutonic igneous body. Leaving the source region under high hydrostatic pressure, and carrying a load of dissolved mineral substance, the solutions travel in the direction of the relief of pressure, which in general would be toward the surface. As they ascend, following permeable avenues of travel through the rock body, the solutions encounter continuously decreasing temperatures and pressures, and when horizons involving appropriate physical conditions are reached, deposition of ore ensues, following the Zonal Theory.

The mineral substances carried in solution are for the most part of such low solubility that the solutions must be regarded as of extreme dilution. It should be clearly recognized, therefore, that vast volumes of solutions must have operated over a very lengthy period of time in order to have produced the deposits of an important mineralized district. Because of this time requirement many concurrent events must transpire during the period of mineralization and many progressive changes may occur in the geological environment.

While in general agreement that mineralizing solutions originate at depth in a plutonic igneous body, geologists apparently are not agreed as to just when in the life cycle of the plutonic body the solutions are expelled; nor as to the immediate source of the solutions, whether from the mass of the plutonic body at large or from some specific phase of the plutonic body, perhaps brought into existence during a late stage of its life cycle as a result of magmatic differentiation. Nor are the permeable channelways followed by the mineralizing solutions in their ascent to the surface ordinarily regarded as other than fortuitous openings, which happily were present at the right time and place.

A close consideration of the events that precede and occur during mineralization discloses a pattern that is repeated with almost monotonous regularity in our mining districts. The regular repetition of the same sequence of events going to form this pattern can hardly be regarded as other than indicating that these events are necessary component parts of a unified process. In this process the stage is first set, the area to be mineralized being actually blocked out and prepared before the beginning of the mineralization.

## IGNEOUS INTRUSION, FAULTING AND MINERALIZATION

The mineralized district usually is an area of limited extent, perhaps an area of only a few square miles. Concentrated within this area occur not only the ore deposits but also igneous intrusives and localized faulting, both of which are closely related in age to the mineralization.

In surrounding areas where mineralization is only sparsely represented or absent, localized faulting of mineral age and igneous intrusives comparable with those of the mineralized district are likewise absent.

So habitually are they associated within the confines of mineralized districts, some interrelationship must necessarily exist between igneous intrusion, faulting and mineralization. The relation between mineralization and igneous activity has long been the subject of intensive study and discussion. In marked contrast, the relation between faulting and igneous activity on the one hand, and between faulting and mineralization on the other, has received relatively scant attention. Yet it should be manifest that the control of the mineralized district is just as dependent on structural disturbance and faulting as is the control of the individual ore body.

If the time relation between igneous activity, faulting and mineralization be evaluated, it will be found to be essentially similar for most mineralized districts. Diagrammatically the time relation may be shown as follows:

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Igneous activity ---------- - - -
Faulting           - - ----------------
Mineralization               ----------
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The mineralization ordinarily follows the igneous activity, frequently being entirely later, but occasionally overlapping in time the closing stages of the igneous history.

The initiation of the faulting may precede the first igneous events or may occur during the early part of the igneous period. Once started, important fault movements continue throughout the time of igneous activity and may persist into the earlier part of the period of mineralization. With passing time, however, the intensity of the fault movements decreases, until during the middle and later stages of the mineralization, they are of only trivial magnitude.

In spite of this, these late fault movements are extremely important, since it is these small movements that are responsible for the fracture openings that control the introduction of the valuable metallic compounds.

The faulting that precedes the mineralization may be of any character. In contrast, some evidence exists that the minor faulting through the later stages of the mineralization is frequently the result of gravitational readjustments.

In individual cases, entirely unrelated periods of igneous activity or of faulting may be superimposed later upon the mineralized district. These do not control but they may modify the features of the mineralization.

The age relations of faulting as just presented are in marked contrast to statements made by Spurr,[1] who, impressed by the lack of mineralization of large faults present in mining camps, concluded that such faults were necessarily postmineral. Actually, faults of large displacement in mineralized districts often show some evidence of having been affected by the mineralization, and occasionally they may locally contain ore bodies. Large faults usually contain an abundant filling of gouge, the result of the movements that have occurred along the fault. This gouge is highly impermeable and relatively inert to the action of the mineralizing solutions, as was clearly recognized by Ransome.[2] In consequence, large faults may persist through a period of mineralization relatively unaffected, while neighboring faults containing but little gouge may become the sites of vein formation.

## THE DEEP ZONE OF MINERALIZATION

Where erosion has been deep enough to expose portions of batholithic bodies, the localization of ore deposits within limited mineralized districts is not too well defined. In such cases, however, as has been so ably shown by Emmons,[3] the deposits occur chiefly in the roof region of the batholith or along its margins. The deposits may occur either in the country rock outside the border of the batholith, or within the batholith itself, but if within the batholith never at any great distance from the batholithic border.

Not infrequently dikes are present whose compositions indicate advanced stages of differentiation of the magma body from which they were derived. Though usually preceding the mineralization, the latest of these dikes may appear within the period of mineralization. Injected progressively along fault fissures cutting the main igneous body, and in general followed by the formation of the vein fissures and the mineralization, these dikes necessarily indicate a considerable lapse of time between the emplacement of the batholith and the start of the mineralization. During this time a considerable portion of the batholith must have become crystallized, advanced stages of differentiation were reached within the body, and recurrent faulting was in progress. Quite certainly at this late stage, the source of the mineralizing solutions could only be from some deeply buried part of the batholith, which is safely removed from view.

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[1] References are at the end of the paper.

### *Grass Valley, California*

The vein deposits at Grass Valley, Calif., occur in and near bodies of granodiorite and other granitic rocks. The nature and distribution of the granitic bodies indicate that erosion has cut down to the roof region of a portion of the Sierra Nevada batholith.

Following solidification of the granodiorite, fissures were formed along which dikes of highly differentiated character were injected, both lamprophyres and aplites. According to Johnston,[4] in general the lamprophyres are the older, and faulting occurred between dike injections. Furthermore, many of the narrower aplite dikes are of very fine grain, and some have chilled borders against the adjacent granodiorite. Apparently, by the time the aplites were injected not only had the granodiorite solidified but fault movements were in progress, and sufficient cooling of the granodiorite had occurred to cause a chilling of the narrower injections of aplite.

Age determinations, using the "helium method," were made by Urry on samples collected by Johnston, with the following results:[4]

| | |
|---|---|
| Granodiorite | 110 ± 5 million years. |
| "Dark gray dike" | 110 ± 7 million years. |
| "Dark gray dike" | 90 ± 7 million years. |
| Aplite dike | 98 ± 4 million years. |

Faulting and the formation of the vein fissures followed the injection of the diachistic dikes, and progressive faulting continued into and throughout the period of mineralization.

In the Empire-Star and the Idaho-Maryland mines, as well as in other deposits of the district, early vein quartz shows repeated fracturing and brecciation with subsequent cementation by younger and younger quartz. The metallic minerals were introduced in a partly overlapping sequence during the later stages of the mineralization. The introduction of the earlier metallic minerals was controlled by fractures cutting the older quartz, whereas the later metallic minerals may transect any of the older minerals along progressively formed fractures. The gold was the latest mineral introduced, partly overlapping but definitely outlasting the galena and chalcopyrite in time.

The fault movements during the later stages of the mineralization appear to have been small, and in most places the effects amount to little more than recurrent microbrecciation of the ore. Postmineral disturbance of the ore is not particularly evident.

The history of events from the granodiorite invasion to the termination of the mineralization that has been outlined for the Grass Valley deposits would apply with almost uncanny accuracy to the deposits along the Mother Lode belt, farther to the south. Diaschistic dikes commonly are present and in places the latest dikes, usually lamprophyres, may be injected within the period of mineralization.

## MINERALIZATION AT MODERATE DEPTH

In mineralized areas where post-mineral erosion has been less intensive, the localization of ore deposits within well-defined districts is the rule. Such camps as Ely, Bingham, Park City and Bisbee are typical of the many examples that might be cited.

Central stocks of limited size are frequently present, but parts of batholithic bodies related in age to the mineralization are not exposed, though such bodies may be presumed to exist at depth. The stocks are commonly controlled in position by earlier faulting. In form, the stocks may be regarded as roughly cylindrical or pipelike, though highly erratic in detail. Swells and pinches may occur, with tongues and apophyses and other projections extending out from the main body in a highly irregular manner. Dikes may be shot out, following fault openings present in the surrounding rock body. Probably in most cases "chonolith" would be a better designation for the intrusive body than "stock." Quite certainly no evidence is available to indicate a general progressive downward enlargement of the stock, with resultant conical form as pictured by Emmons,[3] nor is evidence available to indicate close proximity to the roof of the underlying batholith from which the stock is presumed to be an offshoot.

Later than the solidification of the stock, and cutting it along fracture openings, dikes may appear, of highly differentiated character. These dikes, entirely comparable to those found in mineralized areas exposed by deep erosion, could have originated only in a large intrusive body at depth which had reached a stage of advanced differentiation and solidification.

The ore deposits may occur both in the stock and clustered about its periphery, but seldom at any great distance from the intrusive body. Only a single period of mineralization can be recognized, although more than one period has on occasion been postulated. Not only does evidence for more than one period appear absent, but when examined in detail the sequence of changing events involved in the mineralization is so methodical, so uniform in character, that to duplicate this sequence at closely spaced intervals within the same district appears highly improbable.

The period of mineralization is definitely later than the solidification of the central intrusive, and separated from that event by a sufficient time interval to permit the appearance of highly differentiated types of dikes from depth, and for important fault action to occur. This fault action may involve fracturing and brecciation of the intrusive as well as the creation of faults and vein fissures in various parts of the rock body. Fault movements continue with decreasing vigor throughout the period of mineralization.

Since the mineralizing solutions appear well after the time of solidification of the stock, no source for these solutions can be recognized within the body of the stock itself. The mineralizing solutions must necessarily, therefore, have originated from deep within some underlying plutonic body that was only then undergoing crystallization, presumably the same body from which the dikes of highly differentiated character were shot upward.

Once it is recognized that the source of the mineralizing solutions is not within the body of the stock, but lies deep within an underlying batholithic body, all ideas of direct genetic relationship between the stock and the associated cluster of ore deposits are destroyed. We are forced thereby to recognize that the localization of ore deposits in and about the stock and within the definite confines of the mineralized district is necessarily the result of structural control.

### *Ely, Nevada*

The events leading to the development of the ore deposits near Ely, in eastern Nevada, are completely illustrative of the conditions that have been outlined. A central intrusive body occurs of the composition of monzonite porphyry. Ore deposits occur in and clustered about this body, and because of this association the intrusive is known as the "ore porphyry." Other intrusive bodies of somewhat different age present in this district show no such relationship to ore.

The "ore porphyry" intrusive has been variously regarded as a stock and as a laccolith, but is now known to be a chonolith.[5] Faulting not only controlled the emplacement of this chonolith, but was recurrent throughout the period of igneous invasion and solidification, and continued on into the period of mineralization. The "ore porphyry" was shattered, and faulting developed structures in the surrounding rock body that subsequently were made use of by the mineralization. Lamprophyric dikes of the composition of minette were injected into the solidified "ore porphyry" before the start of the mineralization.[6]

Appearing after solidification, and probably after considerable cooling, of the chonolith, no source can be recognized within the body of the chonolith itself for either the magma of the minette dikes or for the mineralizing solutions.

## Epithermal Mineralization

In many mining camps where shallow or epithermal types of ore deposits occur, the intrusive igneous bodies present may include pipes of small diameter, but more frequently constitute a dike swarm. In some districts only a few dikes may be represented. Regardless of the number present, however, the dikes are found to have an areal distribution roughly similar to that of the mineralization. Furthermore, where dating of the intrusives is possible, they are usually found to have shortly preceded the mineralization, though in some places the later of the dikes may actually be injected within the period of mineralization.

Where well represented, the dikes are injected in a regular sequence of progressively changing composition. The first dikes usually are of intermediate composition, andesites or dacites. With passing time the dikes that are injected are of more and more acidic composition. Finally an acid extreme appears, and shortly following this occur dikes of basic composition, diabases and lamprophyric types.

In districts where dikes are sparsely present, only a few stages in the indicated sequence of composition may be represented, but these appear in the expected order.

The igneous intrusives usually are preceded and accompanied by important fault movements. Early dikes frequently are emplaced along faults that had already undergone considerable displacement. Subsequent movements along these faults may be followed by the injection of later dikes, so that the same fault surface may be followed by two or more successive dikes. Not infrequently veins may later form along these same fault lines. Elsewhere early dikes may be offset by faults, which in turn are occupied by later dike injections.

Important movements frequently continue into the early stages of vein formation. With passing time during the mineralization, however, the fault activity decreases in intensity, so that after the mineralization is well advanced movements of small magnitude are the rule.

To one who has studied a dike swarm within an epithermal district, who then passes beyond the limits of the district and finds such dikes absent and then on entering another mineralized district, 10 or 20 or 50 miles distant, there again encounters a dike assemblage, similar in all respects to that of the first district, only one implication appears possible.

The epithermal district is underlain at no great depth by a stock of moderate diameter, and this in turn must represent a protrusion above the roof of a still deeper batholithic body. The pipes and more abundant dikes of the district, ranging in composition from intermediate to moderately acid, are derived from this stock, the progressively changing composition of the dikes expressing the changes in composition of the stock resulting from mild differentiation. The dikes of extreme acid and those of basic or lamprophyric character, which appear latest in the intrusive sequence, are the counterpart of the similar dikes that appear in the more deeply eroded mineralized districts, and collectively with those, these dikes of highly differentiated character can only be derived from a batholithic body at depth that has reached a stage of advanced differentiation. Appearing later than these highly differentiated dike types, perhaps with some time overlap, the mineralizing solutions also must necessarily be derived from some deep part of this batholithic body.

The localized structural disturbances and faulting, which are coextensive in time and space with the dike swarm and with the mineralization of the epithermal district, can result only from causes and conditions that are dependent on the presence of the underlying stock.

### *Pachuca, Mexico*

The silver deposits of Pachuca,[7,8] Mexico, occur in a thick pile of volcanic flows and interbedded pyroclastics of probable Miocene age. Injected ahead of the mineralization, and cutting all known flows of premineral age, a dike swarm occurs that is largely localized within the known mineralized area. Dacite dikes appeared first, followed by numerous dikes of siliceous, light-colored rhyolite. A few local occurrences of quartz diabase apparently constitute the latest premineral intrusives.

The dacite dikes were injected after an intensively localized fault pattern had already been largely developed. Recurrent faulting continued during the period of dike injection and into the period of vein formation. In a number of instances the same fault was progressively occupied by dacite, by rhyolite, and finally by vein formation.

Fault movements persisted with declining vigor throughout the period of mineralization. Certain of the movements during the later stages of the mineralization, and particularly during the time of introduction of the silver into the veins, appear to have been in the nature of gravitational adjustments occurring within the framework of the previously established fracture pattern.

### *Randsburg, California*

In the Randsburg district,[9] in southern California, mineralization of epithermal character extends over an area of 15 to 20 sq. miles. This area is also the locus of invasion of a dike swarm and of recurrent intensive faulting. The dike invasion was progressive over a considerable period of time, as shown by progressive dike injections along the same fault surfaces, by crosscutting relations between dikes, and in particular by the changing composition of the dikes injected with passing time (Fig. 1).

The majority of the dikes range from andesites to rhyolites. Later than these are types of highly differentiated character. The acid extreme is represented by a holocrystalline rock composed solely of quartz, orthoclase and muscovite. Somewhat later in time are lamprophyric types, usually holocrystalline and frequently diabasic in texture. These were followed by the mineralization.

Faulting started ahead of the first dikes, occurred progressively during the period of dike injection, and persisted with diminishing intensity throughout the period of mineralization. Little in the way of postmineral fault action is evident in the ore deposits.

### *Mojave District, California*

In the near-by Mojave district,* some 50 miles to the westward, comparable history is represented. Dacite flows occur, which are cut by siliceous quartz-porphyry dikes and by a few highly altered lamprophyric dikes, and there is concurrent faulting. Important fault movements persisted into the ensuing period of vein formation, but during the later part of the mineralization, when the metallic minerals were being introduced, the fracture openings progressively developed were of almost microscopic magnitude.

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* Personal observations.

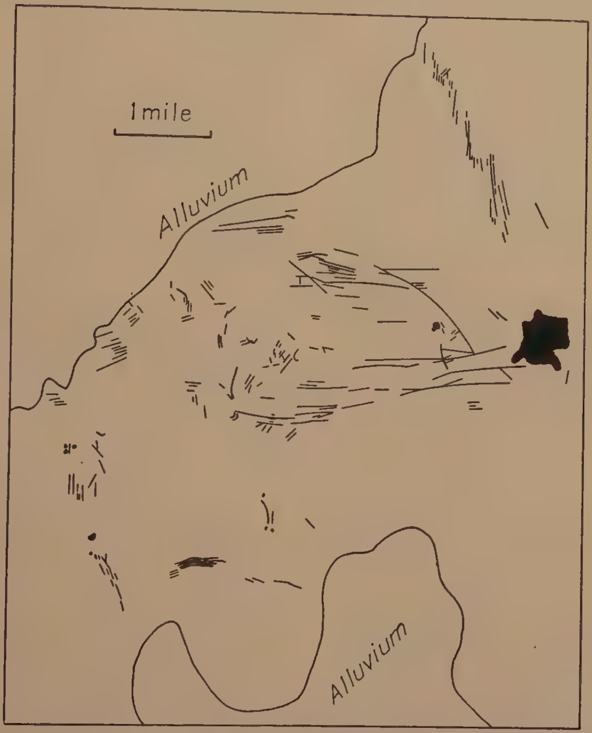

FIG. 1.—IGNEOUS INTRUSIVES RELATED IN AGE TO MINERALIZATION IN RANDSBURG DISTRICT CALIFORNIA.

The principal mineralization extends from the center to the right center of the area shown.

## STRUCTURAL CONTROL OF THE MINERALIZED DISTRICT

Although consideration of the features that characterize mineralized districts has necessarily been brief, and somewhat generalized, several factors stand out as of utmost importance.

Regardless of the depth zone represented, the advent of the mineralizing solutions is always distinctly later in time than the consolidation of the exposed portions of the igneous intrusives with which the ore deposits are associated. An exception is found in occasional dikes of highly differentiated character. The lapse of time involved between these events cannot be evaluated accurately in the light of present information, but such evidence as is available suggests an interval of some magnitude. Not only do important fault movements occur within this interval, but crystallization of the main plutonic body at depth becomes sufficiently advanced that highly differentiated magma facies are generated and shot upward as dikes into the area that subsequently is to become the mineralized district.

Whatever weight may be attached to the "helium-method" age determinations made by Urry[10] on rock specimens from Grass Valley, the results indicate that between the crystallization of that part of the batholithic body there exposed and the beginning of the mineralization, an interval of time elapsed, which was not less than 3,000,000 to 5,000,000 years and which may have been on the order of 8,000,000 to 12,000,000 years.

Once the time intervening between the solidification of the intrusive igneous body and the advent of the associated mineralization becomes clearly recognized, no idea of a direct genetic relationship between the ore deposits and the near-by intrusive body can be longer entertained—nor can a genetic relationship any longer be used to explain the association.

Instead, it must be recognized that the grouping of ore deposits in mineralized districts in close proximity to intrusive igneous bodies is dependent on structural control resulting from forces set in operation by the intrusive body.

### MAINTENANCE OF PERMEABILITY

One important control of the mineralized area is the necessary presence of permeable channelways cutting the rock body. However, the mere presence of such permeable openings is not in itself sufficient; the permeable openings must be accessible to the mineralizing solutions as they originate at depth in the parent plutonic body that has reached an advanced stage of solidification and differentiation. In the absence of such access by the mineralizing solutions, the permeable openings possess no importance.

The permeable openings that pervade the mineralized district are almost entirely of fault origin. In rare cases permeable openings may occur locally, which are of other than fracture nature.

Faults, however, are not inherently permeable. As has been pointed out, faults that have suffered large displacement before the start of the mineralization are frequently filled with fault gouge, which renders them highly impermeable. Favorable conditions of permeability may exist along premineral fault fissures of minor displacement, and along fault fissures that originate at the start of or within the period of mineralization. However, even in these cases the permeability is a function of the relative attitude of the fault surface and the directional stresses present in the rock body. Where the stress has a large component normal to the fault surface, permeability may be reduced to the vanishing point.

The presence at the start of the mineralization of permeable openings in the rock body, and access to these openings by the mineralizing solutions, is still not sufficient to define the mineralized district. The permeable openings that originally exist, and are occupied and used as avenues of travel by the mineralizing solutions, will shortly be filled by mineral products, with the result that the process of mineralization will choke itself off.

For important mineralization to occur, the permeable openings *must be maintained* throughout the period of mineralization. Failure to maintain any particular opening results in the termination of any further mineralization dependent on the presence of that particular opening, and failure to maintain the permeable openings within a district necessarily terminates the mineralization as it affects that district.

Insofar as permeable openings in existence during the mineralization are continuously being filled by mineral products, maintenance of permeability can result only from recurrent fault movements that continue throughout the period of mineralization.

Studies of both ore and barren vein matter show that recurrent fracturing and brecciation, the result of a succession of small fault movements, are apparently universal. This is substantiated by field studies. The writer has yet to encounter a deposit where evidence for such fault history is not written indelibly in the ore.

It is thus apparent that the mineralized district is controlled by: (1) The presence of permeable openings developed before or during the earlier stages of the mineralization by fault activity; (2) access to these openings by the mineralizing solutions originating at depth; (3) maintenance of permeable openings within the district by a succession of fault movements which continue throughout the period of mineralization.

It matters little whether the ore in the particular district is emplaced primarily by deposition in openings or by replacement of portions of the rock body adjoining such openings. In either case vast quantities of solution acting over a very lengthy period of time are required to introduce the necessary amount of new mineral substance, and sealing of the avenues of travel followed by the solutions would inevitably occur unless kept open by fault action.

## Localization of Faulting by Volume Change in the Rock Body

The limitation of the mineralization to an area of localized faulting within which permeability is maintained by continued crustal disturbance appears clear. But what circumstances have controlled the localization of important fault disturbances ahead of the mineralization, and the synchronization of declining fault activity with the mineralization? It is believed that the answer to these questions is found in the volume changes induced in the rock body by the emplacement and presence of the igneous intrusives with which mineralized districts are so characteristically associated.

During the late stages of emplacement of the batholith and of the attendant cupolas and stocks protruded above its roof, and during the subsequent crystallization of these bodies, at least two causes exist for important structural disturbances and faulting: (1) the weakening of the crustal rocks caused by the emplacement of a large body of fluid magma; and (2) the volume change that occurs in the intrusive body as it changes state from fluid to crystalline solid.

## Volume Change on Crystallization of the Batholith

As the large body of fluid magma represented in the batholith is emplaced, the surrounding and overlying regions of the crust would be greatly weakened, to the extent that strain originally present in the crustal rocks would be expected to overcome the strength of the rock body, with consequent failure on a regional scale by folding and by faulting. Weak zones thus produced might well localize the positions of stocks and other bodies protruded above the batholith roof. The associated faulting could be of any type and magnitude, but would be expected commonly to be compressional in nature. Broad problems of regional structure obviously are involved, which will not be followed further in this paper.

Crystallization of the batholith would work progressively from the roof downward, and somewhat more slowly from the outer borders inward. During the earlier stages of solidification, volume decreases resulting from progressive crystallization should result in the gradual slow subsidence of the entire roof region of the batholith. Thus no general cause leading to highly localized strains appears to exist at this time.

During the very late stages of crystallization of the batholith, conditions become quite different. The residual magma still in existence has become highly differentiated in character and occurs in a reservoir of limited extent. Probably a series of such residual magma reservoirs occurs, isolated from each other within the body of the batholith. Withdrawal of support from the overlying rock, the necessary result of the volume shrinkage involved in the continued crystallization, can no longer be compensated by gradual subsidence of broad regions of the roof. Instead, collapse of limited areas above the residual magma reservoirs would be inevitable.

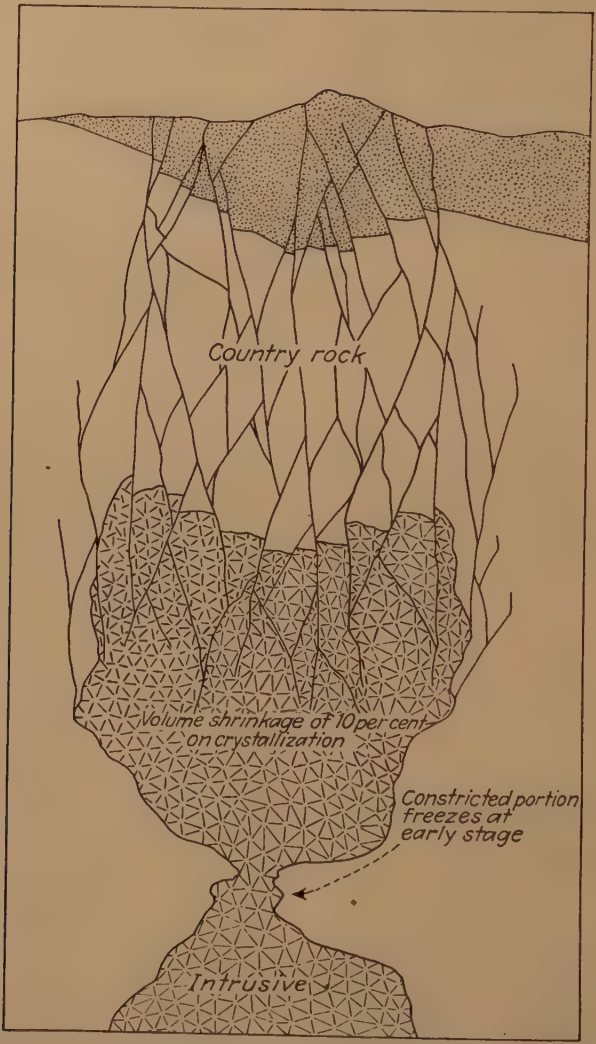
Country rock
Volume shrinkage of 10 per cent on crystallization
Constricted portion freezes at early stage
Intrusive

FIG. 2.—IDEALIZED SECTION OF CHONOLITHIC INTRUSIVE, SHOWING SUBSIDENCE RESULTING FROM VOLUME DECREASE ON CRYSTALLIZATION.

The upper third of the diagram is adapted from a section of the Pilares pipe figured by Locke in 1926.

That such collapse is not merely a theoretical possibility, but actually does occur, is substantiated by the almost universal presence of diaschistic dikes of highly differentiated character in mineralized districts.[11] Such dikes are not limited to districts in the roof region of the batholith, but appear in mineralized districts involving all depth zones up to and including the epithermal. Quite evidently the faults created by collapse extended from the residual magma reservoir practically to the surface.

Not infrequently in mining camps the faults occupied by these diaschistic dikes offset earlier dikes of normal composition, thus restricting the age of this faulting to a time when only the highly differentiated magma of the residual reservoirs was in existence.

## VOLUME CHANGE ON CRYSTALLIZATION OF THE STOCK

The behavior of the stock or chonolith protruded high above the roof of the batholith would be quite different from that of the large underlying parent body. Following emplacement, the body of fluid magma composing the stock would serve to weaken only a limited region in the surrounding and overlying rock body. Strains present in this part of the rock body, if of sufficient intensity, would again be relieved by faulting, which would be largely confined to an annular zone of limited but varying width, surrounding the intrusive body, and to an area of corresponding size overlying the intrusive. This faulting might often make use of older fault surfaces, and as with the older faults, could be of any type and magnitude. Commonly, however, it would be expected to be of compressional nature. Faults developed or active at this time might frequently dominate the fracture pattern of the mineralized district.

The crystallization of the stock would be progressive from the outer borders inward and from the top downward, and at a rate that would vary with the local cross-sectional area of the stock. If the stock were of conical form, showing progressive downward increase in diameter, volume shrinkage as crystallization progressed would be automatically compensated by upward movement of magma from below. Thus, though volume change due to change of state was actually occurring, it could produce no effective result.

If, however, the stock is chonolithic in character, with erratic enlargements and pinches occurring in its cross section at various depths, horizons of minimum cross section would solidify completely while overlying enlarged portions of the intrusive were still partly or largely fluid. Thus cut off from further supply of magma from below, the enlarged part of the intrusive would necessarily crystallize as an individual body. Volume decrease would then occur progressively with crystallization (Fig. 2), possibly causing continued faulting (which might be compressional), in an annular zone surrounding the stock, but in particular promoting progressive faulting and subsidence in the higher portion of the stock and in the region above the stock.

Such progressive subsidence of higher parts of the stock could be accompanied by autobrecciation of the subsiding portion of the rock body. The result would be unsystematized fracturing and shattering with attendant volume increase, structural features so characteristic of the porphyry copper ores and whose presence and extent predetermine the porphyry copper ore body.

In the absence of such autobrecciation, the subsiding ground might move downward within the limits of an encircling, crudely cylindrical fault surface, or within a nest of such surfaces. The result would be structures comparable to those of the Pilares pipe and the Colorado pipe.

The behavior of the stock during crystallization, and the varied attendant structural disturbances that may occur at this time, are thus functions of the form of the individual stock.

## MAGNITUDE OF VOLUME CHANGE RESULTING FROM CHANGE OF STATE

While it cannot be doubted that the magnitude of the volume decrease involved in the change of state from fluid magma to crystalline rock is important, figures accurately representative of this volume change are apparently not available.

However, glass is to be regarded as an undercooled liquid of extreme viscosity.[12] It should follow, therefore, that the relative volumes of a crystalline granitic rock and of equivalent glass at the same temperature should be closely representative of the volume change resulting from crystallization of the magma. The relative volumes are inversely proportional to the specific gravities.

According to Daly,[13] average obsidian has a specific gravity about 10 per cent less than that of average granite under surface conditions. This would indicate a volume decrease in passing from glass to crystalline granite of 10 per cent. This figure manifestly is subject to correction for conditions of elevated temperature and pressure such as would exist in the stock or batholith at the time of crystallization. The effect of pressure in particular is not well understood, but the combined corrections would not be expected to greatly alter the general magnitude of the volume decrease.

Actual measurements made on diabase in the Geophysical Laboratory at Washington[13] show a decrease in specific gravity of 7.19 per cent in the change from crystalline rock to glass at 20°C., and a decrease in specific gravity of 9.9 per cent in the same change at 1200°C. These correspond to decreases in volume of 7.19 and 9.9 per cent, respectively, in changing from glass to crystalline diabase. Again no correction is represented for conditions involving plutonic pressures.

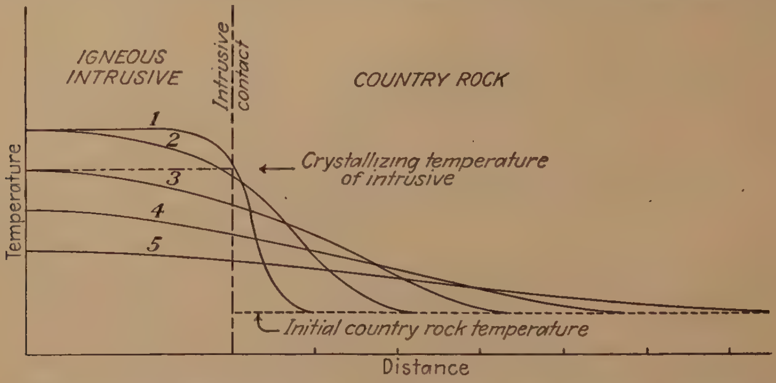

FIG. 3.—DIAGRAMMATIC REPRESENTATION OF CHANGING TEMPERATURE GRADIENTS IN AND ADJACENT TO IGNEOUS INTRUSIVE WITH PASSING TIME FROM 1 TO 5.

Early gradients are steeper; later gradients flatter. Maximum temperatures attained in the country rock are progressively later for points at increased distance from the contact. *Not to scale.*

Tentatively, it would appear reasonable to accept a figure of 10 per cent for the volume decrease involved in the change from fluid magma to crystalline granite. This figure, though not necessarily accurate, is indicative that the magnitude of the volume change resulting from change of state is quite sufficient to account for important structural disturbances.

## HEATING OF THE ROCK BODY

During the time of emplacement and of subsequent consolidation, vast quantities of heat pass from the intrusive to the surrounding and overlying rock body. The available heat is dependent not alone on the size and specific heat of the intrusive, but also involves the latent heat of fusion given off during the crystallization.

The flow of heat into the surrounding rock body results from a thermal gradient involving temperature differences. The maximum temperature attained in the rock body outside the intrusive would occur at the intrusive contact at a time preceding the start of the crystallization. Rock is a very poor conductor of heat, however, so at this time a relatively steep temperature gradient would exist into the surrounding rock body, and relatively low temperatures would still prevail at moderate distances from the intrusive contact (Fig. 3).

With passing time, the temperature at the contact tends to decrease slowly, but because of the outward flow of heat temperatures at moderate distances from the contact are still increasing. Thus, the time of maximum temperature is progressively later for points at greater and greater distance from the intrusive contact.

During the time of temperature increase in the rock body outside the intrusive, thermal expansion would produce an increase in volume. Acting coincidently, thermal metamorphism, by producing minerals of greater density, would induce a a volume decrease. These two volume changes of opposite sign would be compensatory to some degree, and, operating as they do in conjunction with the more important volume changes occurring in the intrusive body, their net effect probably would be of questionable importance.

Although the time of maximum temperature at the contact must precede the start of crystallization of the intrusive, no important decrease in temperature in the zone close to the intrusive can occur until after the crystallization of the intrusive is largely complete, since prior to that time large amounts of heat are being made available from the latent heat of fusion of the intrusive body.

## VOLUME CHANGE DUE TO CHANGING TEMPERATURE

Following completion of the crystallization, the history of the intrusive body and of an annular zone of moderate width surrounding the intrusive body would be one of declining temperature, though at greater distances temperatures might still be slowly increasing. This history of declining temperature would necessarily be accompanied by volume shrinkage, the result of thermal contraction.

This volume change would occur progressively over a long period of time made necessary by the slow rate of heat loss from the rock body. The rate of volume change would vary with time, being greatest at an early period when temperatures were higher and thermal gradients steeper. With passing time as temperatures decreased, the thermal gradients would flatten and the rate of volume change would become less and less. This change in rate of volume change would be further accentuated by the fact that the coefficient of thermal expansion for rock materials varies with temperature, being noticeably greater for higher temperatures within the range involved in this discussion.

The change in rate of the volume change resulting from thermal contraction that occurs with passing time harmonizes closely with the variation in fault activity of the mineralized district. Fault movements of some importance precede and may persist into the early stages of the mineralization. With passing time during the mineralization, however, the fault activity grows weaker and weaker, until in the late stages of the mineralization the movements usually are quite trivial.

The total volume change that occurs would vary with distance from the intrusive. It would be greatest in and close to the intrusive body, where initial temperatures were higher, and would become progressively less at greater distances from the intrusive, owing to the lower maximum temperatures attained at these increased distances. This variation in volume change with distance from the intrusive is probably as effective in promoting structural disturbance as is the actual magnitude of the volume changes (Fig. 4).

Because of this variation, the rock body about a stock would *tend* to be separated into a series of concentric shells by contraction along radial lines, and concurrently differential vertical shortening of successive shells would occur, the maximum shortening being within and close to the stock. These shells, of course, would never be actually developed, since the rock body is never a "homogeneous solid," and the differential stresses would be largely taken up along predefined faults. Thus, while some new fracture structures undoubtedly would be formed, the chief effect of this progressive differential shortening would be to cause continued movements along structures already developed in the rock body.

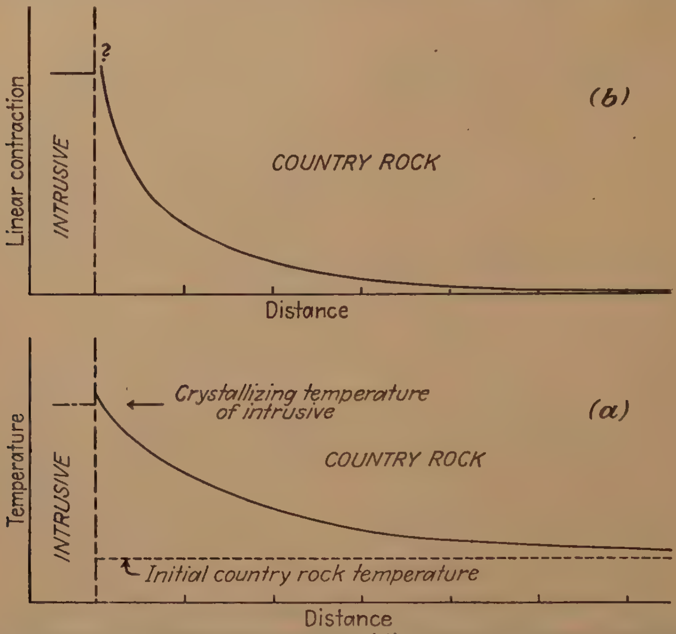

FIG. 4.—DIAGRAMMATIC REPRESENTATION OF ($A$) MAXIMUM TEMPERATURES ATTAINED AT VARYING DISTANCES FROM THE INTRUSIVE CONTACT, AND ($B$) RESULTANT VARIATION IN TOTAL LINEAR THERMAL CONTRACTION WITH DISTANCE FROM THE INTRUSIVE CONTACT. *Not to scale.*

A further cause for variation in the total volume change in the rock body surrounding the stock is found in the change in cross section of the stock that may occur at different horizons. The total heat flowing into the rock body at any horizon would be a function of the cross-sectional area of the stock. Thus, opposite enlarged portions of the stock more heat is available, leading to the heating of a wider zone in the surrounding rock body and to higher temperatures in zones out from the contact. The result here would be important volume change during the cooling period. In contrast, opposite restricted portions of the stock where little heat would be available, the zone of heating would be narrow, and relatively lower temperatures would be attained in zones away from the contact. The result would be volume change of possibly minor importance.

The volume changes occurring about the stock would necessarily be reflected in the rock body above the stock, where successive fault displacements would be generated. The structural history of the epithermal district would thus be largely controlled by events occurring at greater depths.

In the zone close to the batholith, conditions in general would be comparable to those existing in the vicinity of the stock. Successive shells of rock, conforming to the isogeothermal surfaces near the batholithic border, would contract differentially, with resultant strain and fracture. Conditions would appear particularly favorable for the localization of differential strain in the vicinity of cupolas and other prominences of the batholithic surface.

### MAGNITUDE OF VOLUME CHANGE RESULTING FROM THERMAL CONTRACTION

The thermal expansion of rocks in the range up to 300°C. has been subject to considerable investigation, but above that temperature few accurate measurements are available. Data summarized from the International Critical Tables[14] yield averages for the coefficient of linear thermal expansion for different types of rock as shown in Table 1.

TABLE 1.—*Average Values, Coefficient of Linear Thermal Expansion*
($\mu \times 10^{-6}$)

| Rock | Number of Samples | 0° or 20°C. to 100°C. | 100°C. to 200°C. | 200°C. to 300°C. |
|---|---|---|---|---|
| Limestone | 4 | 11.0 | 17.1 | 22.2 |
| Marble | 6 | 10.5 | 19.5 | 23.0 |
| Quartzite | 1 | 16. | 20. | 20. |
| Sandstone | 2 | 11.0 | 15.5 | 19. |
| Slate | 1 | 9.4 | 9.7 | |
| Granite | 1 | 9. | 14. | 20. |
| Granite | 3 | 7.0 | 14. | |
| Diabase | 1 | 6.3 | 9. | 12. |

More recent work, carried out by Griffith[15] at the Iowa State College, gives much higher values. Thus, for the interval from room temperature to 212°F., the average of the values of the coefficient of linear thermal expansion obtained for 16 granites is $44.3 \times 10^{-6}$, with a range from 19 to 66. These values are much higher than any reported by other investigators. Numerous determinations by Griffith of the coefficient for other types of rock likewise are much higher in value than are published elsewhere.

Table 1 shows that the coefficient of thermal expansion apparently increases with temperature for all rocks. One set of determinations of the coefficient for diabase, which is cited by Daly,[18] shows the coefficient of thermal expansion to increase materially up to 600°C., above which the coefficient decreases with further increase of temperature. The average coefficient up to 300°C. is 9.2; that from 300°C. to 600°C. is 25.7, an increase of almost three times.

Experimental data obtained by Kozu[16] involving the determination of the coefficient of linear thermal expansion for five granites, when averaged, give the following:

| | | |
|---|---|---|
| Range | 20°C. to 300°C. | $13.2 \times 10^{-6}$ |
| | 150°C. to 300°C. | $18.0 \times 10^{-6}$ |
| | 300°C. to 600°C. | $43.5 \times 10^{-6}$ |

The value of the coefficient for the lower temperature range agrees closely with the values listed for granite in the International Critical Tables. Furthermore, the ratio of the values for the two temperature ranges is similar to the ratio for diabase for these same ranges as shown by the figures cited by Daly.[18]

The averages obtained from Kozu's figures are therefore used as the basis for the statements that follow.

For temperatures above 300°C., granite would show a change in linear dimension of 4.35 ft. per 1000 ft. for each change of 100°C. in temperature. For the range 300°C. to 150°C., the linear change would be 1.80 ft. per 1000 ft. for each 100°C. change in temperature.

Thus, if it be assumed that the granitic rock crystallizes at about 700°C., and that the mineralization starts at 300°C. and terminates at 150°C., a stock 5000 ft. in diameter would decrease in dimension by about 87 ft. in the interval between its crystallization and the start of the mineralization. A further shrinkage in dimension of 13.5 ft. would occur during the time of the mineralization. Thus, between the crystallization of the stock and the termination of the mineralization at the temperatures assumed, thermal contraction would produce a volume decrease of almost 6 per cent. If this shrinkage be compensated by the downward settling of wedge-shaped blocks, important displacements along a succession of faults could be accommodated by the amount of shrinkage indicated.

Although complete data for sediments are lacking, the coefficients for sediments as given in the International Critical Tables suggest that the volume change within the rock body immediately outside the intrusive is probably somewhat greater than within the body of granitic rock.

From the magnitude of the figures cited, whatever their exact accuracy, it seems clear that volume change resulting from thermal contraction in the rock body is an important factor, and one that is quite competent to produce the fault movements that closely precede the mineralization, and which may extend into the earlier stages of the mineralization. Occurring continuously, as they do, the decreasing volume changes that take place as the rock body becomes cooler would be peculiarly adapted as the cause of the recurrent minor movements, which are manifest through all the later stages of the mineralization.

## CONCLUSIONS

The association of ore deposits and intrusive igneous bodies which is so characteristic of mineralized districts cannot be regarded as resulting from any genetic relationship between the mineralization and adjacent portions of the intrusive body. Instead, the association is recognized as resulting from structural control of the mineralized district.

The emplacement and presence of the intrusive body are responsible for structural disturbances, which not only prepare the district ahead of the mineralization but which also maintain the permeability that is so necessary for the continuance of the mineralization and the formation of commercial ore bodies in the mineralized district. The permeable channelways traveled by the mineralizing solutions from their source at depth are likewise occasioned by events in the igneous history.

Igneous invasion, structural disturbance and mineralization must be regarded as component parts of a normal igneous cycle that has gone to completion.

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## DISCUSSION

W. R. WADE*—I can agree with Dr. Hulin that the igneous stocks and batholiths in intruding the rocks weakened them and caused breaks and faults, and that the shrinkage of these intrusive granitic bodies opened many fissures that later became the channels for ore solutions and the loci of ore veins or breccia-type ore bodies. But I can not agree that the sole function of the intrusives was to prepare the ground mechanically for the deposition of ore, and that the ore is not generically related to the granitic intrusives.

In the case of the deposits of Tintic, Utah, there are distinct ore channels running northerly for distances of several miles from the igneous monzonite stock at Silver City. These ore channels show a zonal arrangement as they extend north out in the limestone beds from the igneous stock. At the southern end the ore is low-grade pyrite with gold and copper. A mile or so farther north the ore channels contain lead-zinc ore bodies, while still farther north lead-silver predominates, and at their extreme northern end they are filled chiefly with chalcedony and calcite. These ore channels resemble pre-existing cold water-formed caves in the limestone and are so considered by R. T. Walker, who believes these caves were filled by ore which came in solution from the south. The igneous stock on cooling gave off large quantities of hot mineral solutions which traveled through these limestone caves. The stock, when sufficiently cool, fractured and was mineralized with low-grade pyritic gold quartz veins.

It is difficult to conceive how the igneous stock at one end could have produced caves in the limestone, which are similar in structure to the caves like the Mammoth Cave in Kentucky, but it is very easy to see how hot gases and mineral waters given off by the cooling magma of the stock would find these open channels and use them as a means of escape. Here, plainly, we have a case where a granitic mass did not mechanically prepare the region for the ore bodies, but did, as far as we can see, furnish the minerals to fill the pre-existing caves.

Dr. Hulin cites Pilares in his ideal cross section. The writer operated Pilares for several years and is very familiar with that mine. At Pilares the ore-bearing oval area, which is similar in shape to a volcanic crater, has ore bodies around the elliptical rim and at irregular intervals in the center of the area. These ore bodies all have a practically vertical dip. The Pilares crater—if we may call it such—is surrounded by monzonite. The rocks in the crater are andesite below the 700 level and latite above. The greatest ore bodies are at the two ends of the oval and at the contact with the monzonite outer wall.

There is a definite generic association between the monzonite and the ore. The copper and iron sulphides very clearly rose along the monzonite contact. Moreover in connecting to the new Pilares shaft we drove our cross cuts on each level 300 ft into the rimming monzonite. We cut many little veinlets carrying pyrite and copper. Also, specks of sulphides occur through the intrusive monzonite.

Dr. Hulin mentions Bisbee to prove his point that the igneous stock, or chonolith (as he calls it) did not furnish the mineral, but by shrinkage prepared the channels in both the igneous and the invaded rocks, which in this case were chiefly limestone. The igneous stock, a monzonite porphyry mass called Sacramento Hill, has been regarded by most geologists as the source of the copper; especially as the mass itself is copper-bearing and twenty million tons of it were mined as ore.

We have an ideal picture book granitic baby batholith here at Marysville. It is intruded into Helena limestone and Empire shale. A portion of the top of the granite (quartz diorite) is exposed in the Marysville basin with the invaded rocks forming the hills around the basin. The limestone and shale are baked to hornstone for about one-half mile around the intrusive.

Gold veins occur in the shell of the granite and in the hornstone. The pay ore never extends more than 500 ft into the granite, either downward from the hornstone above or horizontally where the contact is vertical, as it is on the eastern side. The quartz veins continue into the granite, but barren. The veins in the hornstone parallel to the granite contact, such as the Drumlummon, carry pay ore to depths of 1200 ft or more, but these pay veins are within one or two hundred feet of the granite contact. Tongues or irregular shoots of granite extend out and form one wall of almost every pay shoot. When both walls are hornstone and the granite is not within a few feet, then the vein will have quartz assaying perhaps 50 cents or less. But as soon as a tongue of granite cuts into or beside the fissure the ore is good for $15.00 or better.

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* Drumlummon Mine, Marysville, Mont.

In the case of the Empire shoot (Drumlummon mine), the vein extended northeast away from the granite-hornstone contact at about a 45° angle as a barren quartz-filled fissure for 1000 ft. Then a tongue of granite formed the footwall and an ore shoot assaying $25.00 per ton began and continued for the next 600 ft to a short distance past the granite. Then the quartz became barren again.

In summation I agree with Dr. Hulin that the granitic intrusives in our western mining camps cause:

1. Mechanical openings, faults and fissures by their invasion of the original rocks.

2. That the contraction of these igneous rocks on cooling leave the invaded rocks above unsupported, thereby causing settling faults and fissures.

3. That the granitic type intrusive rocks shrink enough on cooling to fracture their own crust.

I do not agree with Dr. Hulin when he says these intrusives are generically the source of the ore bodies in most of our western mining camps.

J. H. Farrell*—The theoretical discussion and descriptions of districts in this paper appear to imply that the theory covers the formation of practically all ore deposits associated with igneous activity except magmatic segregation and contact metamorphic types. Certain facts observable in many districts are disregarded in giving emphasis to the importance of structural control, with the result that the presentation of the author's views seems rather one-sided.

Professor Hulin offers the following principal points:

1. Ore deposits are associated with intrusive rocks.

2. Ore deposition takes place some time after the intrusion of the main igneous mass.

3. Mineralizing solutions must originate at great depth rather than *in the igneous rocks near the zone of deposition.*

4. Ore deposits are formed in fracture systems caused by the intrusion. The fracturing takes place during the invasion by the intruding rock and during the cooling period.

5. Continuing small fault movements keep the country rock and the vein channels open for mineral-bearing solutions which must act *in great volume and over a long period of time.*

The author repeatedly emphasizes the presumably deep-seated origin of "mineralizing solutions," entirely neglecting the action of some igneous bodies as carriers of the ore minerals, along with the "vein dike" theory of ore deposition, neither of which ideas can be entirely disregarded in any discussion of ore deposits associated with igneous rocks. He says in his second paragraph: "Certainly no source for these solutions can be found in nearby portions of the intrusive igneous body." And in his conclusions he states: "The association of ore deposits and intrusive igneous bodies which is so characteristic of mineralized districts cannot be regarded as resulting from any genetic relationship between the mineralization and adjacent portions of the intrusive body." This is a very broad statement not qualified by reference to the fact that some igneous rocks contain disseminated sulphides or other ore minerals in sufficient quantities to make ore bodies either in their original degree of concentration or after being subjected to secondary enrichment. Nor are contact metamorphic deposits noted as an exception although their mineral content is very evidently derived in part from adjacent intrusive masses.

Professor Hulin was doubtless thinking of the higher-zone ore deposits and probably assumed that these others are so evidently of different origin that they need not be mentioned. The point that the writer would like to make here is that if the facts regarding the origin of such ore deposits in or near igneous masses are admitted it is difficult to deny the agency of other igneous masses as "mineralizers" or "ore carriers."

The author's line of reasoning is that mineralizing solutions must have come from great depth because they appear later than the "differentiated dike types" which cut the main mass of solidified intrusive rock. He says (page 42, paragraph 1), "These dikes of highly differentiated character can only be derived from a batholithic body at depth that has reached a stage of advanced differentiation." It is difficult to follow Professor Hulin in this reasoning, which is highly theoretical since we really know very little about the conditions under which magmatic differentiation takes place or its relation to ore formation. If we go along to his conclusion we find that it means that the existence of igneous formations in or near the zone of mineralization, that is, in the vicinity of ore bodies, is without significance except as the intrusive masses may have produced fracturing or structure favorable to ore deposition.

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* Los Altos, Calif.

Consequently, it entirely upsets the concept that certain volcanic flows, dikes, stocks or other igneous masses may have had an original mineral content which has been concentrated into ore deposits either in the rock masses or adjacent to them. Many geologists, engineers and prospectors who have been primarily concerned with finding ore, are going to find it difficult to give up this idea. It is true that many igneous rock masses show no evidence of original mineral content and are clearly not connected with ore deposition, although structural conditions in their vicinity and rock characteristics may be quite similar to those which appear to be connected with the mineralization of the district. This supports the idea that some particular intrusive bodies are ore carriers while others are not.

As in so much geological theory, the truth may lie in a compromise of extreme views. The "vein dike" theory, so ably developed by J. E. Spurr as a result of exceptionally wide field observation, does away with "mineralizing solutions" as commonly conceived and covers certain ore occurrences, but by no means as many as he would have us believe. The theory of mineralizing solutions originating at depth, possibly not as great depth as suggested by Professor Hulin, applies to other deposits. In certain districts there are extensive vein systems cutting thick series of volcanic flows, as at Mogollon, N. Mex., and near Ouray, Colo. These fit in quite well with the ideas he outlines, except in regard to the length of the period of mineralization. The mineral content of the veins was not derived from the adjacent volcanic rock, and the veins were productive to considerable depth, continuing below the productive zone. On the other hand, there was no extensive alteration of the wall rocks, or even of the rock fragments enclosed in the veins, such as would be expected as a result of long-continued action by hot mineral-bearing solutions.

Somewhere in his discussion the author might have said that in areas where there are extensive fracture systems and many diaschistic dikes of highly differentiated types, the veins are likely to have had origin from mineralizing solutions emanating from batholithic igneous masses, and may be expected to persist in depth. The practical prospector (not afflicted by scientific gobbledygook*) would express the same idea by saying that where lots of dikes of different kinds and many faults crisscross the country there are likely to be "true fisher" (fissure) veins—good ones.

The writer differs with Professor Hulin in his insistence that there can be no genetic connection between ore deposits and adjacent intrusive rocks, presumably in most mineral districts. This difference of opinion is based partly upon theoretical considerations and in part upon field observation covering many ore deposits associated with igneous rocks of various types. To begin with, mineral districts are by no means common in areas marginal to great igneous masses, though that is where they are usually found. They must, therefore, primarily be localized because of some unusual condition such as an exceptional concentration or segregation of metal in the underlying magmatic mass or what amounts to the same thing, the existence of a local pocket or magmatic reservoir containing an unusual proportion of material forced up from the metallic core of the earth. Little can be learned from surface examination about this first necessary step in the localization of mineral districts because of the great changes which take place before the deeper zones are exposed by erosion, but neither lobes, nor stocks, nor any other form of intrusive mass, with attendant shattering and faulting, will have associated ore bodies without the initial concentration of metals.

It is reasonable to suppose that any intrusive body originating in or near such a metal-charged portion of the magmatic reservoir will carry a greater or lesser metallic content with it even as far as the surface of the earth. There is evidence of this in the primary metallic minerals found in stocks and dikes of granitic rock and even in volcanic flows. There is evidence that differentiation takes place in relatively small intrusive masses, in which may be seen segregations of quartz or the basic minerals merging into dikes that cut not only the parent rock but also adjacent rocks. Certain types of quartz veins carrying gold and other metallic minerals develop in this process of differentiation and are very similar to some of the dikes. Sometimes the veins are older, sometimes the dikes. Possibly some veins which formed at comparatively low temperature were fed from local centers as an end stage of differentiation and cooling. The fact that some veins are later than the "differentiated dike types" does not prove that their formation was caused by solutions coming from great depth.

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* Gobbledygook: excessively technical phraseology.

Veins which cannot be followed into underlying formations are occasionally found in rhyolite and other volcanic flow rocks. They range in size from small stringers to veins several feet in width and hundreds of feet long. The simplest explanation of their origin is that they were formed from material carried by the flow. The contrary view is that the mineral-bearing solutions came up from below leaving no trace of their courses and were simply arrested in the formation in which the vein is found. This is much more difficult to believe than the first explanation.

Ore minerals occur as original constituents of granitic textured rocks, sometimes in sufficient amount to make ore. Then come the contact metamorphic deposits, sometimes with adjacent veins closely connected in origin in which the characteristic minerals are clearly derived from the contact zone. Segregations of silicious material carrying sulphides or other ore minerals are found merging into stringer zones or veins in granitic rock masses of relatively small size. The Engels copper deposit, in Plumas County, California, was of this type, the sulphide minerals, pyrite, cupriferous pyrite, and chalcopyrite were associated with a considerable percentage of magnetite and igneous quartz. The mine workings exposed the roots or beginnings of a vein structure and the ore minerals were evidently primary constituents of the igneous mass. With the surplus quartz from the country rock, quartz diorite, the vein minerals marked progressive stages of vein formation along a line of fracturing, beginning with irregular bodies of vein material occurring at random in quite a wide zone; from this condition developed poorly defined stringer zones, then parallel veins with connecting veinlets, and finally a quartz-sulphide vein with inclusions of country rock. The lateral limits were not well-defined walls, yet the deposit had a distinctly vein-like form. There was no indication of mineralizing solutions having passed along this vein zone for very long periods. On the contrary, the impression was very definite that the vein material had been forced into an incipient shear or fault zone as the rock mass cooled, and that the vein formation was merely an end stage of the period of solidification.

Between these limits, that is, the magmatic or deep zone type of ore body and the surface flow type, are found other deposits which make a rather definite sequence. They are not "vein dikes" by any means but were formed by minerals carried from the deep zone of fusion and magmatic differentiation by intrusive igneous masses. One of the districts described by Professor Hulin, the Randsburg district of California, offers certain evidence supporting the idea that a dike or other intrusive mass may bring its mineral content from depth to be deposited under epithermal conditions. There the Yellow Aster mine had several ore bodies in a thick dike of granite. They did not extend into the underlying schists and only penetrated the hanging wall schist to a slight extent. They were found at irregular intervals down the dip of the big dike occupying transverse fractures, and included well-defined veins, extensive stockwork or stringer zone deposits, and areas of granite in which native gold was found usually with microscopic blobs of quartz or in small quartz stringers—incipient segregations of vein material. There was no general alteration of the granite to suggest the passage of great quantities of hot mineralizing solutions for a long period of time. In fact, some of the ore bodies were remarkably fresh looking granite and showed only minor traces of alteration under the microscope. The Yellow Aster ore bodies seem to illustrate a clear case of vein origin within an intrusive mass. Such conditions suggest the possible explanation of the occurrence of other ore bodies which have formed more or less erratically in masses of igneous rock.

Professor Hulin states categorically: "The mineral substances carried in solution are for the most part of such low solubility that the solutions must be regarded as of extreme dilution" (page 37, paragraph 3). He deduces that therefore "vast volumes of solutions must have operated over a very lengthy period of time in order to have produced the deposits of an important mineralized district." The time element thus established is quite important to the development of his ideas regarding ore deposition.

As a matter of fact, we do not know much about the nature of "solutions" at great depth in igneous bodies, either as to their chemical composition or solvent powers. One thing we do know for sure is that they must be quite different from any solutions that we can see at work at the depths to which we have penetrated the earth's crust. However, we may reasonably ask whether we always find evidence in the field indicating the action of "vast volumes of solutions" in connection with important ore deposits. The answer is that sometimes we find much alteration attributable to hydrothermal action in the wall rock near ore deposits and also in the country rock at long distances from the ore zone. In many cases there is very little such alteration in or near a vein with great continuity both in depth and length. Such a vein channel ought to give exceptional opportunity for the circulation of solutions from depth yet the wall rocks are easily recognizable as to type, and fragments within the vein are equally fresh. Areas of extreme hydrothermal alteration showing little or no mineral or metal value, are common in and near mineral districts. It seems that such action may be the end phase of the process of ore deposition, spreading quite widely in the shallower zone, with little significance except as a general indicator of conditions favoring the occurrence of ore deposits.

The study of the origin of ores involves so many factors and so many still unknown and complex quantities, that it is always necessary to be on guard against too much theory. Only by checking constantly against field observation can workable compromises be reached between theory and fact. That is why papers like this one are always valuable and stimulating even if one cannot agree with all of the author's ideas and conclusions.

# Certain Ore Shoots on Warped Fault Planes

By W. H. Emmons,* Member A.I.M.E.

(New York Meeting, February 1942)

## Abstract

Many mineral veins occupy faults, and movements on certain warped fault planes have resulted in openings. On normal faults the steeper parts have the widest openings, and on reverse faults the flatter parts have the widest openings. Theoretically the width of the opening is the sine of the angle of deflection times the amount of movement on the fault plane, but actually the widths of some of the openings are probably less because of readjustments of walls along joints or of other readjustments.

The first treatment of the subject seems to have been in a book on mineral veins by John Leithart, published in Newcastle, England, in 1838. Later the subject was treated in various texts published in Germany, in England, and elsewhere. The subject was not given much prominence in the earlier American literature on economic geology, but has interested several workers in that field in the past decade. In this paper examples of districts showing ore shoots on warped faults, with relations noted, are described.

Fault veins of the three depth classes of veins are noted. Many epithermal veins are in normal faults, and commonly the larger ore shoots are in their steeper portions. Many mesothermal veins are in reverse faults and many others are in normal faults. The flatter parts of veins in reverse faults are commonly wider than the steeper parts, although they are not always as rich as the steeper segments of the veins. There are not many hypothermal veins in faults of measured displacement.

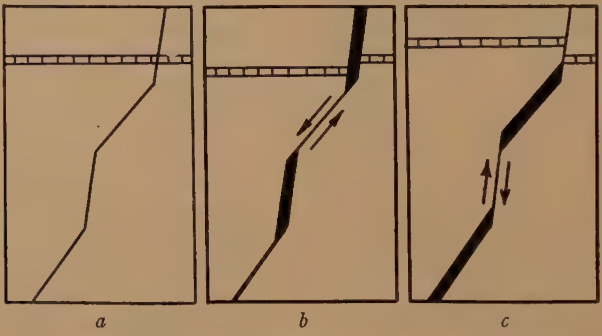

Fig. 1.—Effects of faulting on widths of openings.
*a.* A fissure that changes dip with depth.
*b.* The same after normal faulting. The steepest parts of the fault have widest openings.
*c.* The same after reverse faulting. The steepest parts of the fault have narrowest openings.

## Introduction

When faults are followed along dip or strike, many of them are found to have curving or warped surfaces, and cross sections through them in most directions show sinuous or waving intersections. Openings are formed where one wall moves away from the other (Figs. 1, 2). Where there is a bend in the fault, movement in any directions except along the axis of the bend will

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Manuscript received at the office of the Institute Dec. 8, 1941. Issued as TP 1545 in Mining Technology, January 1943.

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tend to create an opening. In normal faults wider parts of fissure openings should be found in steeper segments (Fig. 1). In reverse faults the wider parts should be found in the segments of the faults that have the lowest dips.

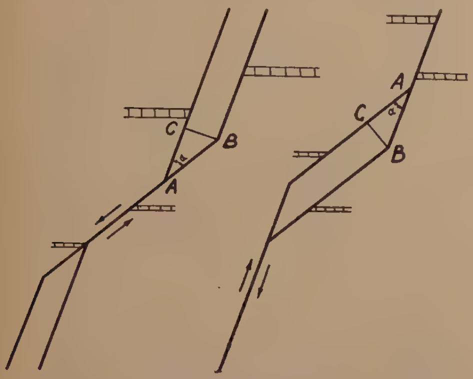

FIG. 2.—RELATIONS OF WIDTHS OF OPENINGS TO AMOUNTS OF DISPLACEMENTS ALONG FAULTS AND TO AMOUNTS OF CHANGES OF DIPS.

*a.* Width of opening formed by normal fault. Angle $CAB = \alpha'$ if $AB = 1$ and $\sin CAB =$ width of opening.

*b.* Width of opening formed by reverse fault. Angle $CAB = \alpha'$ if $AB = 1$ and $\sin CAB =$ width of opening.

The width of the opening depends upon the angle of deflection; that is, the difference between the dip of the fault above and below the place where the dip changes, and the amount of movement (Fig. 2) on the fault plane. If $\alpha$ is the angle representing the difference of dips

$$\sin \alpha = \frac{\text{width of opening}}{\text{displacement along fault}}$$

and if the displacement = 1, the width of opening is $\sin \alpha$.

If the displacement along the fault is 1 (Fig. 2) and the difference in dip is 30°, the width of opening is ½. With a change of angle of dip of one degree and with a displacement of 100 ft., an opening of 1.74 ft. would be formed; and with 100 ft. displacement and a difference of 5°, an opening of 8.71 ft. would result (Fig. 3).

Horizontal deflections also cause openings. This paper aims to test the theory outlined above; namely, that openings and ore veins are commonly wider on the steeper parts of normal faults and on the flatter parts of reverse faults than in their other segments.

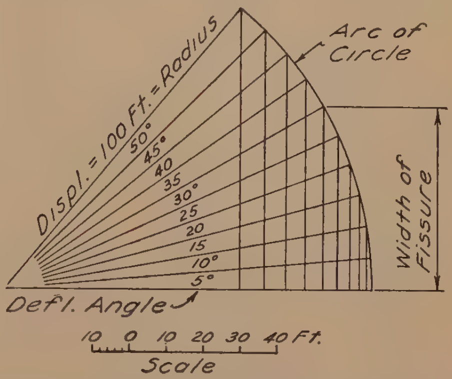

FIG. 3.—WIDTH OF FISSURE FOR DIFFERENT ANGLES OF DEFLECTION ON FAULT PLANE WHERE DISPLACEMENT IS 100 FEET.

### *Historical Review*

It has been said that Werner treated the theory stated above in his lectures, but the first published statement of it seems to have been in a book by Leithart[36] issued in 1838. Leithart was a self-educated miner. He says that he never attended any school except Sunday School, and that he lived his entire life in North England. In his 86-page book on mineral veins he presents some 60 figures. These in general show faulted Paleozoic beds, mainly those lying nearly flat just above the Great Whin sill and crossed by faults with small throws. The beds consist of limestone, sandstone, and shale, and where the fault fissures pass from shale to the stronger limestone or sandstone, they bend so as to cross the limestone with shorter paths. Leithart does not explain the reason for the bends of the veins, but not one of the figures shows an exception to the character of the bends.

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[36] References are at the end of the paper.

Von Cotta,[71] of Freiberg Academy in 1859, 1870, treated the subject rather fully, particularly in the second edition of his text, and so did Beck, in 1903.[5] Von Groddeck,[72] of Clausthal, discussed the relationship briefly in 1879, and Stelzner-Bergeat,[67] also of Clausthal, dealt with it more fully in 1904. Posepny[52] does not mention the theory in his paper read before the American Institute of Mining Engineers in 1892. Beyschlag, Krusch, and Vogt,[8] whose text was published in 1909, gives little attention to the subject.

Phillips and Louis[51] show the effects of movement along a warped fracture and illustrate with diagrams the irregular widths of openings formed by the process. Their text refers to the treatment of the subject by Leithart. Rastall,[60] in a text issued in 1923, discusses the subject of ore shoots on faults and presents an illustration of a normal fault with ore shoots, and also cites an example of one that shows wide parts of the vein on steeper segments of a normal fault.

The early American literature on veins does not give the theory much prominence, although several of the most active workers in the early days of the development of economic geology in the United States had been students at Freiberg, where the theory had been developed. Kemp[32] does not illustrate it in his text, nor does Ries.[62] Lindgren[37] mentions but does not illustrate the subject in any of his four editions. S. F. Emmons and F. L. Ransome, who made outstanding contributions to the origin of vein formation, seem to have ignored the subject.

In the first edition of my text[16] on ore deposits (1918) I illustrate the manner of formation of wide spaces along warped faults, but the subject is discussed very briefly and not very convincingly. In a later edition of the book more attention is given to the subject, but it is not sufficiently well supported by actual examples.[17]

Purington was the first in this country to discuss the subject convincingly (1897). He stated[55] that both the Smuggler and Virginius veins of Telluride, Colo., fill openings in faults of small throw, but he presented no maps exhibiting such features. The map of the Telluride quadrangle by Cross and Purington (1899)[54] shows a few faults with small throws and many veins, but none showing offsets of mineralized fractures.

The Camp Bird vein lies a mile or two east of the Telluride quadrangle. Spurr[65] mapped the vein in detail and described it in 1925. The vein is in a fault and the movement is essentially horizontal. At the four places where the strike of the vein changes, the four ore shoots of the mine were formed.

In 1929 Hulin[30] discussed the subject in general and supported the theory with a plan and section of the Sunnyside mine, near Silverton, Colorado, and a plan of the Arevalo mine, Pachuca, Mexico This is one of the most convincing papers on the subject that has appeared.

Ferguson and Gannett[21] in 1932 noted that the ore shoots of the Sixteen to One lode, Alleghany, Calif., are deposited in a reverse fault, and that they are widest and richest at places of less than average dip.

The same year Burbank[10] discussed the subject in reporting on the Bonanza district, Colorado.

Behre, Scott and Banfield (1937) ascribed[6,7] the various widths of the Wisconsin zinc veins to movements on reverse faults.

Lovering, in his paper on Montezuma, Colo. (1935),[42] presented maps illustrating the widening of veins due to faulting, and in his paper on the Boulder County[43] tungsten belt, Colorado (1940), he discussed the theory and presented a map illustrating the effect of changes of strike in forming ore shoots of the Cold Spring vein.

Recently Newhouse[49] has treated the geometry and mechanics of the subject very fully and used the structure-contour method for defining the attitudes of fault-fissure veins. Farmin[20] mentioned the subject in discussing the origin of gold quartz veins of Grass Valley, California.

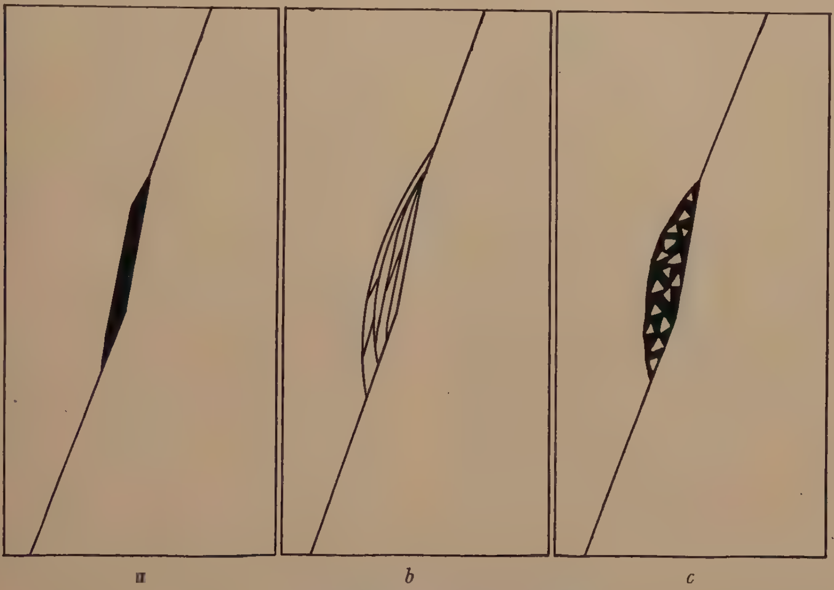


FIG. 4.—ORE SHOOTS ON NORMAL FAULTS.

*a.* Ore shoot formed by filling single opening at place where vein becomes steeper downdip of normal fault.

*b.* Ore shoot formed by filling many nearly parallel openings near place where vein becomes steeper downdip of normal fault.

*c.* Ore shoot formed by cementation of breccia where vein becomes steeper downdip of normal fault.

In a paper by McKinstry,[46] recently issued, the structural features of ore-bearing fissures are treated and ore shoots on warped fault planes are given considerable attention. Analyses of several districts in Australia are presented.

### OPENINGS ON WARPED FAULTS

As a result of movement downdip along warped faults, any one of the following situations may be expected:

1. A single opening at the change in attitude of the fault. (Fig. 4*a*).

2. A group of openings such as a group of fractures nearly parallel to the fault (Fig. 4*b*), or a sheeted zone along the segment where the dip changes.

3. A brecciation of the wall rock near the change of attitude of the fault (Fig. 4*c*).

Any one of these three kinds of structures might result in a considerable widening of a vein or in the formation of an ore shoot in a vein elsewhere unworkable. If the veins fill openings, the shattered walls, as well as the main fault opening, may provide the spaces to be filled. If deposition is by replacement, the fractured material is more readily replaced because metal-bearing solutions have more ready access to the broken rocks.

### DIFFICULTIES IN APPLICATION OF THEORY

A theory is stated here that is almost axiomatic in its simplicity, yet many leading students of ore deposits since 1890 either rejected or ignored it. This seems to imply weaknesses in the theory that require explanation. Some probable explanations are stated below:

1. Probably almost every geologist mapping an area with complicated faulting has sketched parts of his map on transparent paper, cut it apart along a fault and slid it back and forth to aid his visualization of the structure before faulting. On warped faults of considerable displacement and deflection, many of the openings required by the theory are very much greater (Fig. 3) than are observed in the field and greater than deposits commonly filling openings along the faults. It is a natural inference that other faults, perhaps some not exposed, have caused the walls to approach each other more closely and thus have reduced the sizes of most of the larger openings, or that readjustments of joint blocks of the walls have reduced the widths of openings along the faults. I am not satisfied with the explanation myself, because I have not noted evidence of sufficient adjustments of joint blocks to account for certain closures, but it is the best explanation I can suggest.

2. Not only are many openings along faults smaller than would be expected from the theory, but the number of examples of fissure veins illustrating the theory was few, and maps and cross sections indicating such openings, until the last two decades, were almost wanting. Until the present century the mining companies seldom had their own geologic staffs to map the workings day by day. Government parties visited a field periodically and often found large parts of the workings caved and inaccessible. In recent years, also, the practice of detailed mapping and recording observations has greatly improved.

3. Many contacts are gradational. That is true not only of sedimentary formations, but of certain igneous ones such as tuff beds, flows, and so forth. Thus one may know from friction breccia, fault gouge, striae, grooves, and other features that a vein occupies a fault but may not be able to estimate the amount of the displacement. This is important when it is recalled that a change of dip of 5° on a fault with only 20 ft. of displacement may result in an opening of 1.74 ft. Such an opening, when filled with ore accompanied by a little replacement of the walls, might localize a considerable ore shoot on a fault.

4. The illustrations of the theory showing wedgelike openings (Fig. 1) resulting from displacement on faults with change of dip or strike are perhaps unfortunate. Such illustrations were used by Von Cotta, Bergeaut, Hulin, Spurr, W. H. Emmons, and others. They are effective pedagogically. The method of illustration (Fig. 1) is too greatly idealized. I have never seen a vein that shows the features illustrated so sharply and so far as I know few if any have been described. Purington's illustration[55] is better. He shows rounded changes of dip or strike of faults, and actual examples illustrating the theory in this paper show the same feature—curving rather than angular changes of dip and strike of veins.

5. It is possible that sheeted zones (Fig. 4*b*) and brecciated masses (Fig. 4*c*) on the steeper parts of normal faults and on the flatter parts of reverse faults have not been recognized as examples, yet these should be considered true examples illustrating the theory stated, since many of them doubtless are due to fractures or to brecciation of blocks above the open spaces.

6. Many replacement deposits that lie along fissure veins are at places where the wall rocks are shattered, sheeted, or otherwise made permeable to the replacing solutions. Thus, even in replacement deposits, the wider parts of the deposits should be expected to conform to the theory reviewed here. The great replacements along a reverse fault should be expected in the hanging wall of the flatter segments of the fault.

7. Many ore shoots on faults, as elsewhere, were localized by more than one favorable structural feature. They include intersections of veins with favorable beds, with anticlines, with wall rocks favorable

for replacement, with brittle and therefore more easily fractured wall rocks, and other features. These and many others may obscure the relations of the ore shoots to flat or steep segments of fault fissure veins.

8. There are examples of fault fissure veins where wider parts are in the flatter segment of a normal fault. The ore shoots of some such veins may be explained reasonably. Thus, in the Oro mine, near the El Oro shaft, Breckenridge, Colo., the Siam vein, in a normal fault with a throw down-dip of the section of about 18 ft., has a flatter segment which is wider than the steeper one, but the wider segment is in brittle porphyry and the narrow one is in shale.[41]

## COMPARISON OF METALLIZED FAULT FISSURES

The classification of veins[38] according to the physical conditions attending their deposition, which in turn depend largely on the depths of their formation, is convenient for the comparison and study of fault fissure veins. Table 1 outlines some of the features of hypothermal, mesothermal and epithermal veins.[48] I include with the epithermal veins such lead-zinc deposits of the Wisconsin and North England type as are related to faults. Whether these deposits were formed by meteoric waters or by thermal waters that had moved far from their sources (telethermal deposits) is not important for the present discussion. The veins on faults with measured throw are found in the mesothermal and epithermal groups.

TABLE 1.—*Comparison of Certain Features of Vein Deposits*[a]

| Veins | In Normal Faults | In Reverse Faults | Breccia | Replacement | Metal Zoning |
|---|---|---|---|---|---|
| EPI-THERMAL | Many veins are in normal faults | Veins in reverse faults much less common than in normal faults | Very common | Not very important economically in most districts | Generally not definite. Telescoping common |
| MESO-THERMAL | Veins in and along normal faults common | Many veins in and along reverse faults | Common | Common, but not economically important in all mesothermal deposits | Show zoning commonly and more clearly than other classes |
| HYPO-THERMAL | Veins in normal faults rare but not unknown | Veins common in reverse faults but most of them of small throw | Less common except in pipelike and closely related deposits | Replacement very common. Few hypothermal veins are simple fillings | Relatively rare in pre-Cambrian deposits, well shown in some areas of Paleozoic mineralization (England) |

[a] See W. H. Newhouse.[48]

## EPITHERMAL AND TELETHERMAL DEPOSITS

Epithermal veins are formed at relatively shallow depths. More large openings generally are present than in regions of deeper deposition, and in many districts faults are closely spaced. Fault blocks are moved more readily in the shallow zone because they are under lighter loads. Hydrothermal alteration of wall rock is commonly intense—at places it extends over hundreds or thousands of feet—yet ore formed by replacement of wall rocks is generally very subordinate to that which fills spaces. In general, openings are plentiful near the surface when pressures are low. Metals in the rising fluids may readily be disposed of by filling, and thus replacement of rock by ore is less pronounced than in regions of mesothermal and hypothermal deposits. Rock fragments cemented by ore and ore fragments cemented by a second or a third generation of ore are common.

Veins associated with normal faults are much more numerous than those in reverse faults. Therefore, since most of the mineralized faults are normal, ore shoots in warped faults should be expected in the steeper parts of the veins. There are many examples. They include the great ore shoot known as "Big Cave" stope of the Last Chance mine, Amethyst vein, Creede, Colo.; the Block 14 stope, Commodore mine of the same district; the Montgomery Shoshone ore body of Rhyolite, Nev.; the ore shoots of the Washington vein, Sunnyside mine of Silverton, Colo., and many others.

### *Creede, Colorado*

The Creede[19] district is in Mineral County, southwestern Colorado, near the headwaters of the Rio Grande River and on the east side of the San Juan Mountains. The district has produced more than $42,000,000, nearly all silver and gold, the greater part of it from the Amethyst lode. The district lies in the great volcanic region of the San Juan Mountains. Eleven or more formations are known, each consisting of one or more flows of rhyolite, latite, andesite, basalt, or of tuffs, with some intrusive sheets and dikes.

The deposits (Fig. 5) are epithermal fissure veins filling normal faults. The Amethyst vein is followed on strike with slight interruption for 3 miles. The Amethyst fault system is at least 7 miles long. The vertical displacement of the fault is several hundred feet. The Amethyst lode is on a normal fault, strongly mineralized for about 9000 ft. along strike. In this part of the fault system the fault is nearly straight (Fig. 5*a*). Downdip on block 14, Commodore mine (Fig. 5*b*) between levels 2 and 3, the fault becomes steeper, its dip increasing from about 68° to 73°. In this region fractures are formed in the hanging wall and the ore body becomes notably wider. This part of the lode system was the most productive of the Commodore mine, yielding a large part of the production of the mine, which is 16 million ounces of silver with considerable lead and gold.

The Last Chance mine shows on an even grander scale an ore shoot of the lode that is probably due to steepening of the fault. This mine joins the Amethyst mine to the south and it worked a little more than a full claim on the Amethyst lode, about 1800 ft. of its strike length. From the outcrop of the Amethyst vein to the Nelson adit level the vertical distance is 1200 ft. This part of the lode is about 20 per cent of the well-mineralized part of the Amethyst fault, yet it produced about $19,000,000 above the Nelson adit, or nearly half of the production of the Amethyst fault. A great part of this production was derived from a stope between levels 2 and 6. The average dip of the fault is 65°, but between levels 2 and 6 it dips 70° to 75°. In the hanging wall of the fault six veins were worked (Fig. 5*c*). The section is downdip. Above level 6, and corresponding to the steep part of the fault on levels 3, 4 and 5, there was a great stope called the "Big Cave." (Compare Figs. 4*b* and 4*c*.) The ore was so much fractured and the ground so unstable that much of it was mined without blasting, milling the dirt from the bottom on levels 5 and 6 by drawing it out by gravity from as high as level 2. Considerable ore was thus mined and after a little sorting it was sent to the smelters. When I mapped the geology[18] of the mine in 1911, I was able to see the ground above and below and at the two ends of the stope, but the "Big Cave" stope itself was inaccessible. Those who worked the stope said that at places it was 100 ft. wide. This is noteworthy, since the usual width of stopes on the Amethyst fault is from 3 to 5 feet.

### *Camp Bird, Colorado*

The Camp Bird mine[65,53] is about 7 miles southwest of Ouray, Colo., and a short distance east of the well-known mineral district of Telluride. It is an area of Tertiary lavas and its deposits belong to the later

Tertiary[9] period of metallization of the San Juan region. The lavas lie nearly flat and are strongly fissured but the tangential movements along the fissures are very small and the maps of the Telluride district issued by the United States Geological Survey do not show the veins as faults. The deposits are great epithermal veins. The Camp Bird vein (Fig. 6) cuts through the lavas (Fig. 6*d*) but most of the mineralization is in the andesite explosion breccia and first andesite sheet (1, 2, 3, Fig. 6*c*). The

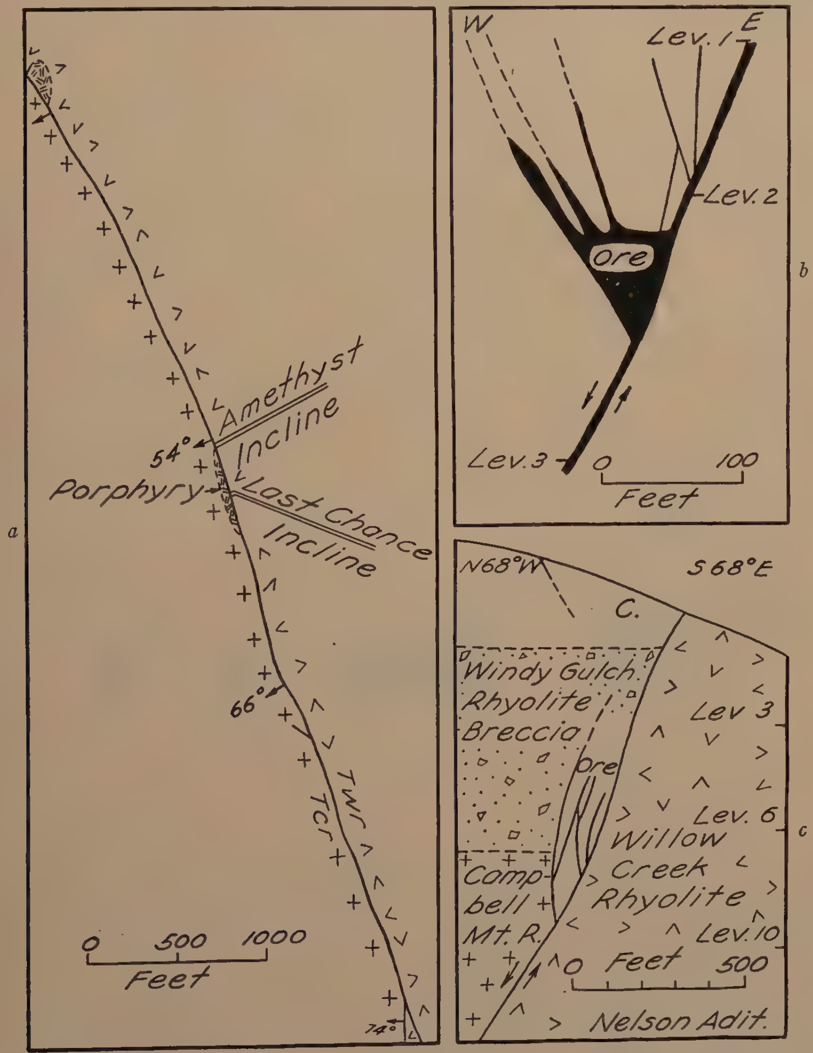


FIG. 5.—MAPS AND CROSS SECTION OF AMETHYST VEIN, CREEDE, COLORADO.

*a*. Part of Nelson adit showing Amethyst fault vein.

*b*. Cross section of block 14, Commodore mine, levels 1 to 3; shows hanging-wall veins and widening of Amethyst vein where fault becomes steeper.

*c*. Cross section of Amethyst vein, Last Chance mine, showing Big Cave ore shoot. Between level 2 and level 6 the ore body removed was at places 100 ft. wide and some of the ore was brecciated and poorly cemented so that it was drawn from the stopes without shooting it.

(All from W. H. Emmons and E. S. Larson, U. S. Geol. Survey.)

average strike of the vein is S.75°E., but at places it strikes nearly east and at four places about S.55°E. to S.69°E (Fig. 6*a*). It was at these four places, where the strike changes, that the ore shoots were formed. As shown by Spurr, with a change of strike of 20° a movement of the north wall of 15 ft. to the east would yield an opening about 5 ft. wide (Fig. 6*b*).

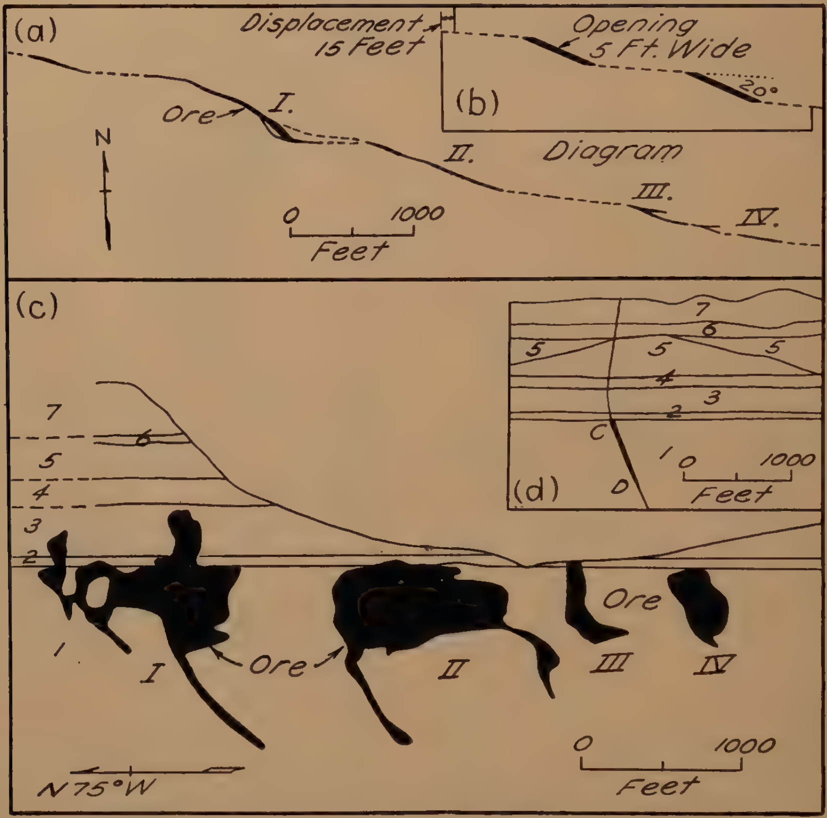


FIG. 6.—CAMP BIRD VEIN.

*a.* Plan of Camp Bird vein, second level of Camp Bird mine (after Spurr). Owing to changes of strike a horizontal movement of north wall to the east causes openings at places of changes. I, Discovery shoot; II, Bluebird shoot; III, Gertrude shoot; IV, Hancock shoot.

*b.* Mechanism of formation of openings of *a.* Strike changes of 20° and a movement of 15 ft. produces an opening of 5 ft. (after Spurr).

*c.* Projection of ore shoots (black) I, II, III, IV on plane of Camp Bird vein (after Spurr). These are marked same as in *a.* Rocks are: (1) andesite explosion breccias, (2) andesite, first sheet, (3) andesite explosion breccias, (4) andesite, second sheet, (5) andesite breccias and flows, (6) andesite, (7) rhyolite.

*d.* Cross section showing Camp Bird vein. *CD* shows ore shoot, which dips 65°S. (after Spurr).

### *Silverton, Colorado*

The Sunnyside[30,56] mine is about 7 miles north northeast of Silverton, Colo. The Washington vein occupies a normal fault, which strikes southeast. From the surface to a depth of about 1000 ft., the dip is about 75°. At a depth of about 350 ft., it becomes steeper and also wider and richer.

In the mine ore shoots are developed where the strike of a fissure changes (Fig. 7*b*). A movement of the north block of the fissure to the west would draw the fissure walls apart, thus making wider the openings between them.

### *Boulder County, Colorado*

The Boulder County tungsten belt[24,27,43,44] is about 4 miles west of Boulder, Colo., and extends farther westward about 10 miles. Since 1900 it has produced a little more than 20,000 tons of tungsten concentrates, with about 60 per cent $WO_3$. The district lies in an area of pre-Cambrian schists, gneisses, and granites, which are intruded by Laramide hornblende monzonite porphyry and by biotite monzonite porphyry and associated latite intrusion breccias of Tertiary age which, according to Lovering, probably represent the source magma from which the tungsten ores were derived. Nearly all of the deposits fill fault fissures with small throws. These fissures have relatively large horizontal components of movement and many with sinuous strike lines present openings through which the metal-bearing solutions rose and deposited ore. Ore shoots are found near places where the strikes of the lodes change. The Cold Spring mine (Fig. 8) illustrates this. The lode strikes east-northeast and dips steeply north. One walking with the forward-moving wall to his right should expect ore where the course of the vein swings to the left, as illustrated by Fig. 8, in which there are four ore shoots, each at places where the vein strike changes. In general the mineralized faults are normal and ore shoots appear on their steeper parts. According to Lovering, reverse faults are uncommon and valuable production is rare from flatter parts of veins.

### *Bullfrog, Nevada*

In the Bullfrog[59] district, about 85 miles south of Tonopah, a thick series of Tertiary lava flows, chiefly rhyolites and basalts, with an aggregate thickness of 6000 ft., is crossed by normal faults, several of which are mineralized to form epithermal veins. Of these the Montgomery-Shoshone fault (Fig. 9) strikes about N.50°E. and has an average dip of about 67°NW. It is a normal fault with a downthrow of about 1500 ft. The district has produced about $3,000,000 gold-silver ore, nearly all of it from the Montgomery-Shoshone mine (Fig. 9*a*). The footwall is rhyolite (No. 10) and the hanging wall is rhyolite (No. 16) under a flow of basalt. The usual dip of the Montgomery-Shoshone fault is as stated, about 67°NW., but in the lower levels of the mine it is 69° or more. From the surface to a depth of about 200 ft. the dip of the fault is very steep, at places almost vertical. The chief deposit of ore is found where the fault and lode are steepest. There the main ore body is 50 ft. wide. Veins and fractures nearly parallel to the main fault are found in the footwall block. Much of the ore consists of shattered rhyolite and breccia cemented by quartz, calcite, and pyrite (Fig. 4*c*).

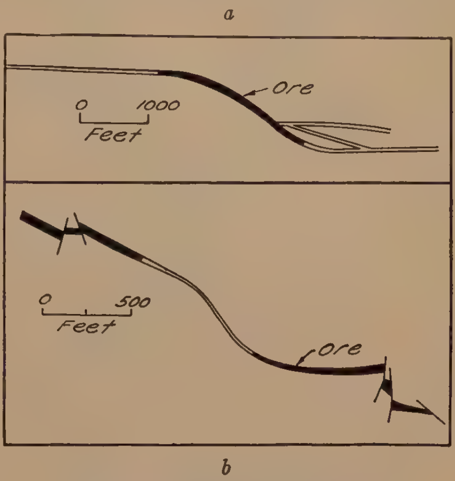


FIG. 7.—WIDENING OF VEINS.

*a*. Plan of part of a level of Arevalo mine, Pachuca, Mexico.

*b*. Plan of part of Sunnyside mine, near Silverton, Colorado. Widening of veins from narrow fissures is explained by horizontal movements.

(Both from Hulin.)

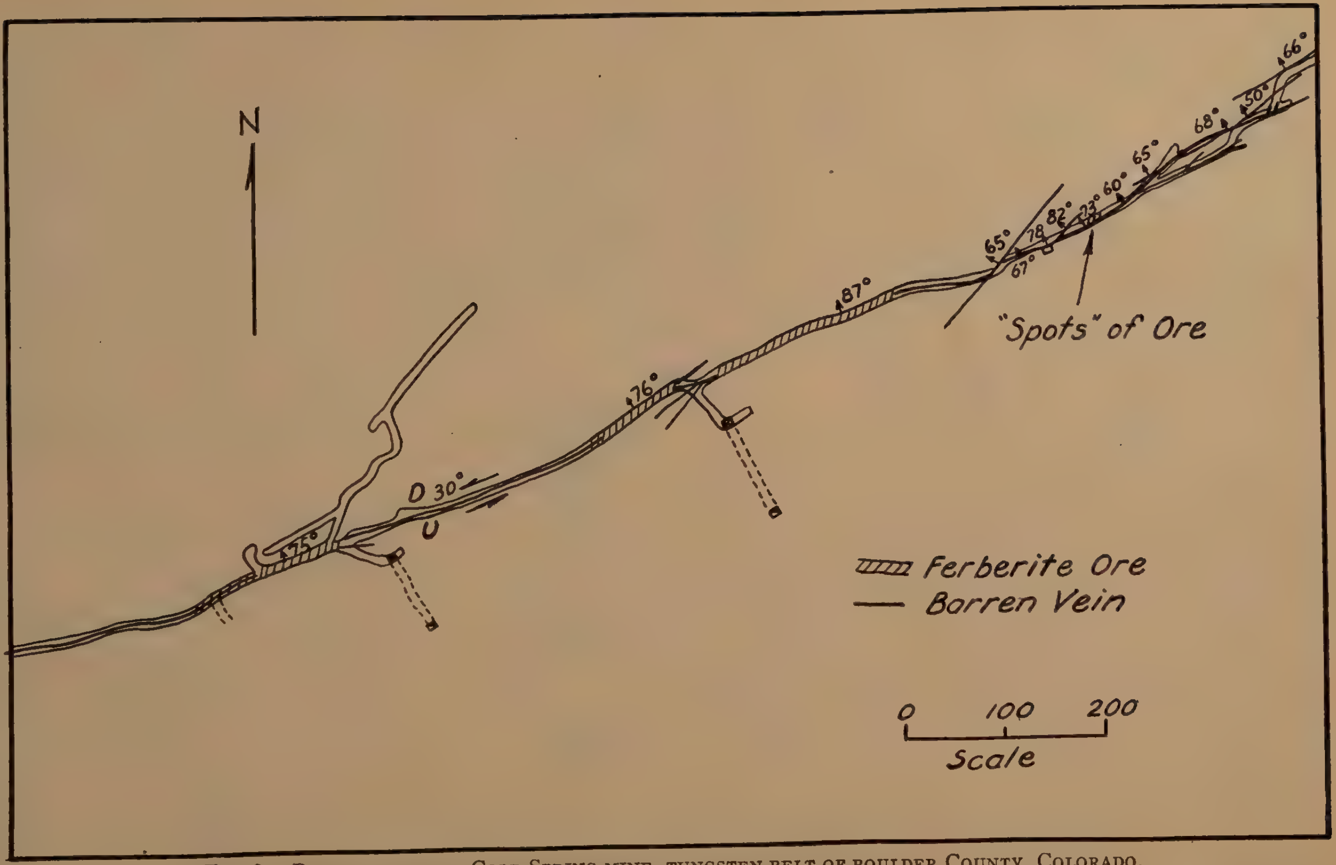


FIG. 8.—PART OF LEVEL 5, COLD SPRING MINE, TUNGSTEN BELT OF BOULDER COUNTY, COLORADO. (after T. S. Lovering, U. S. Geol. Survey.)

One walking with the forward-moving wall to his right should expect ore shoots where the vein swings left, as illustrated by each of the four ore shoots.

### *Pachuca, Mexico*

At the Arevalo mine,[30] Pachuca, Mexico (Fig. 7*a*), a fault of great length is generally barren, but at a place where the strike changes 30° there is an ore shoot wider than the average that is followed on strike for about 2400 feet.

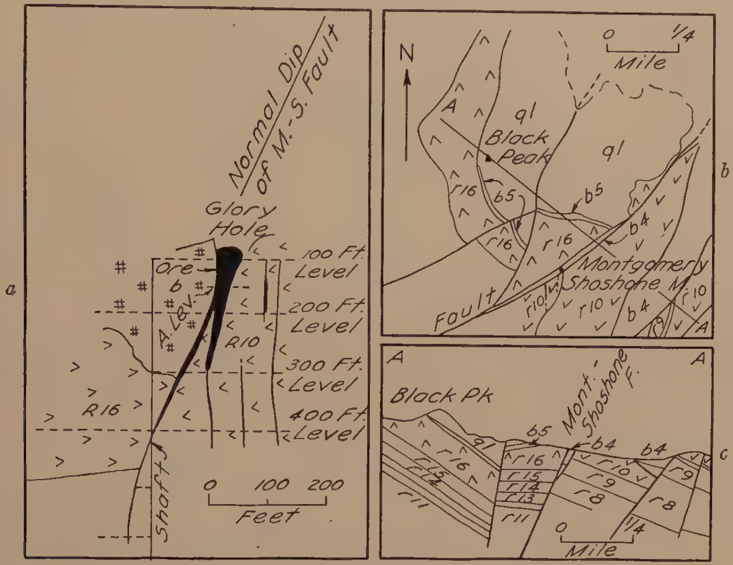


FIG. 9.—MAP AND SECTIONS BULLFROG DISTRICT, NEVADA.
*a.* Geologic cross section of Montgomery Shoshone mine, Bullfrog, Nevada.
*b.* Geological map of part of Bullfrog district, showing position of Montgomery Shoshone mine.
*c.* Geological cross section of part of Bullfrog district, showing position of Montgomery Shoshone fault.
(All from T. L. Ransome, W. H. Emmons and G. H. Garrey, U. S. Geol. Survey.)

### *Upper Mississippi Valley Lead-zinc District*

The Upper Mississippi Valley lead-zinc district of Wisconsin, Ill., and Iowa[1,7,23,25] has produced more than $150,000,000 lead and zinc and still has a small annual production. The country is an area of Paleozoic sedimentary rocks with low regional dips westward away from the Wisconsin arch. Although the rocks are nearly flat lying, there are small, gentle, structural basins and anticlines. The ore deposits generally lie in the structural basins, commonly on their flanks not far above their centers. Several faults with throws from 5 to 15 ft. cross the beds, according to Behre, Scott and Banfield.[7] The ore consists chiefly of marcasite, pyrite, sphalerite, "wurtzite," galena and calcite, and a little barite and quartz. The deposits include: (1) vertical "crevices," (2) flat-lying deposits, between beds, (3) fissures crossing the beds (pitches), (4) ore-cementing breccia blocks in these or other fractured zones, (5) a little lead and zinc ore, not of great economic importance, replacing the oil rock of the Guttenberg formation and the underlying shales of the Platteville formation. Most of the ore fills fissures. Of the faults noted in several of the mines, many are mineralized reverse faults. As shown by Behre, Scott and Banfield, the veins in the parts of the reverse faults with low dips are wider than elsewhere (Fig. 10*c*).

### *Wolfclough Mine, England*

In Yorkshire and neighboring counties of northern England there are a considerable number of lead deposits, some of which have been worked in recent years. The Wolfclough lode, about 2½ miles west of Bolts Burn, lies in a series of sandstones, limestones and shales, with low dips that are faulted by a normal fault having a throw of about 20 ft. The fault is filled by zinc-lead ore and is developed to a depth of about 350 ft. The average dip, as shown on Fig. 10*b*, is about 62° but at places it is as low as 20° and at other places as steep as 82°. The steeper parts of the vein are notably wider than the parts with lower dips.[64]

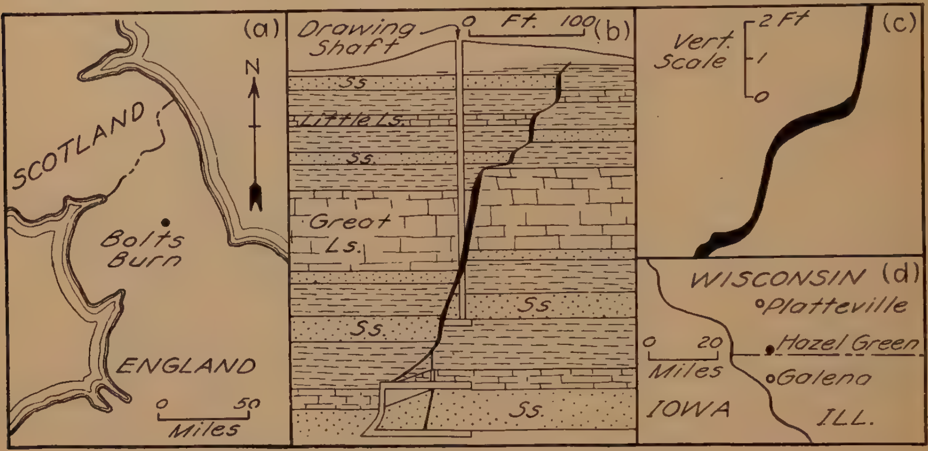


FIG. 10.—WOLFCLOUGH VEIN AND DEPOSIT AT HAZEL GREEN.

*a.* Index map showing location of Bolts Burn Wolfclough, 2½ miles west of Bolts Burn.

*b.* Cross section of Wolfclough vein (after S. Smith). The deposit is a zigzag normal fault vein with narrow flats; steeper parts of veins are widest.

*c.* Cross section of deposit at Crawford mine near Hazel Green, Wisconsin (after Behre, Scott and Banfield). The deposit is a zigzag reverse fault vein with wide flats. The steeper parts of the vein are narrowest.

*d.* Index map showing location of Hazel Green, Wisconsin.

### MESOTHERMAL DEPOSITS

Mesothermal deposits are found in and along both normal and reverse faults. The veins associated with reverse faults include many of those of the Mother Lode, Grass Valley, and Alleghany districts, California. Many of those of the Mother Lode are wider, though usually of lower grade along the parts of the veins that have the lowest dips. The Sixteen to One deposits, which, as noted, are on a sinuous reverse fault, are wider and also more profitable on the flatter segments. Some of the lodes of the Coeur d'Alene, Idaho, are deposited along reverse faults of small throw.

At Butte, Mont., the deposits of the Blue vein (northwest) system are in faults of measured displacement. In these faults the hanging walls moved downward, but there was a greater element of horizontal movement than of vertical movement. The lodes of the Blue vein system carry valuable ore shoots.

### *Alleghany, California*

The Alleghany[21,49] district, California, 16 miles north of Nevada City, is an area of the Calaveras (Carboniferous) system series of rocks, which includes the Tightner formation, consisting chiefly of amphibole schists that probably are the metamorphosed products of andesitic lavas or tuffs and other igneous rocks. The Tightner formation is intruded by serpentine dikes and sills and both by granodiorite and other igneous masses.

The principal deposits are mesothermal veins that strike nearly north, dip east from 30° to 60° and occupy reverse faults, generally of moderate throw. The Sixteen to One mine produced more than $9,000,000, or nearly half the gold of the district. According to Ferguson and Gannett,[21] in the productive portions of the lode the dips are flatter than average. This feature of the Sixteen to One vein is shown on the cross sections in Fig. 11. The displacement along the Sixteen to One reverse fault as measured on the vein is from 500 to 900 feet.

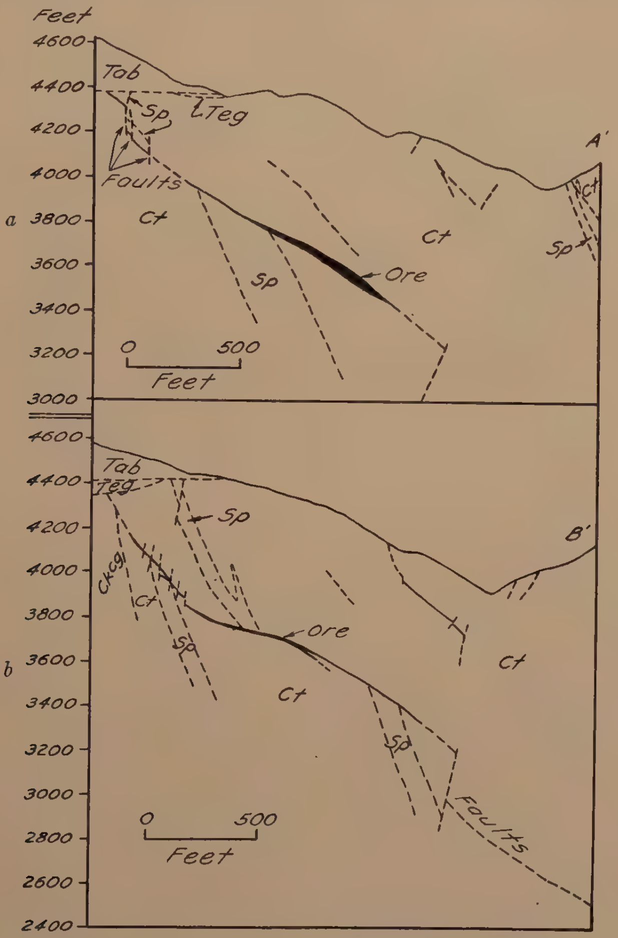


FIG. 11.—CROSS SECTIONS OF SIXTEEN TO ONE VEIN NEAR ALLEGHANY, CALIFORNIA (after Ferguson and Gannett, U. S. Geol. Survey).

Orientation of sections (see plate 10 and p. 106, ref. 21). The vein is a reverse fault with widest portions of ore (black) in flatter parts of vein. *Tab*, andesite breccia; *Teg*, early gravel; *Sp*, serpentine; *Ct*, Tightner formation (Mississippian).

### *Grass Valley, California*

The Grass Valley[20,29,31,33,39,40] district is on the west slope of the Sierra Nevada, about 50 miles north-northeast of Sacramento and 20 miles southwest of Alleghany. A granitic cupola 5 miles long intrudes pre-

Cretaceous rocks. The deposits are great gold quartz veins that lie mainly parallel to the long north-south axis of the cupola and dip at rather low angles to the east and to the west, forming a great conjugated system of veins. Veins of the two groups that dip into the stock are most persistent. The veins seem to be chiefly in reverse faults and where displacement could be measured by dikes that were crossed and displaced Johnston and Cloos found that the maximum movement was about 20 feet.

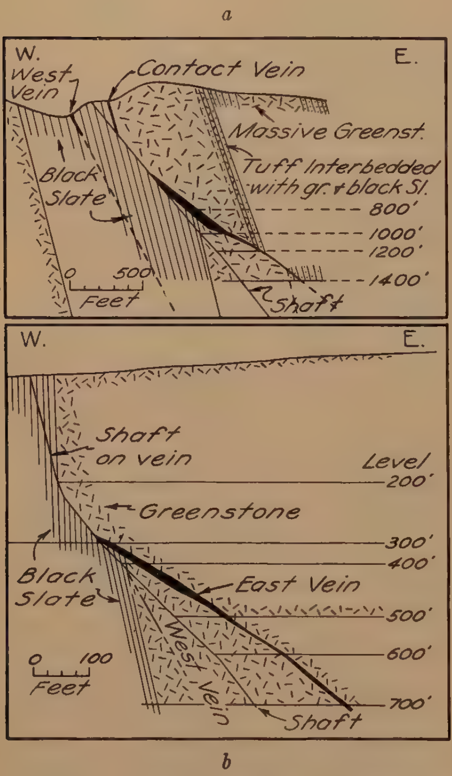


FIG. 12.—CROSS SECTIONS OF MINES OF MOTHER LODE, CALIFORNIA, SHOWING DECREASE OF DIP OF VEINS AS THEY PASS FROM SLATE TO GREENSTONE.

*a*. Geological cross section through Patton shaft, Keystone mine (after Knopf, U. S. Geol. Survey).

*b*. Geological cross section of Original Amadore mine (after Knopf, U. S. Geol. Survey).

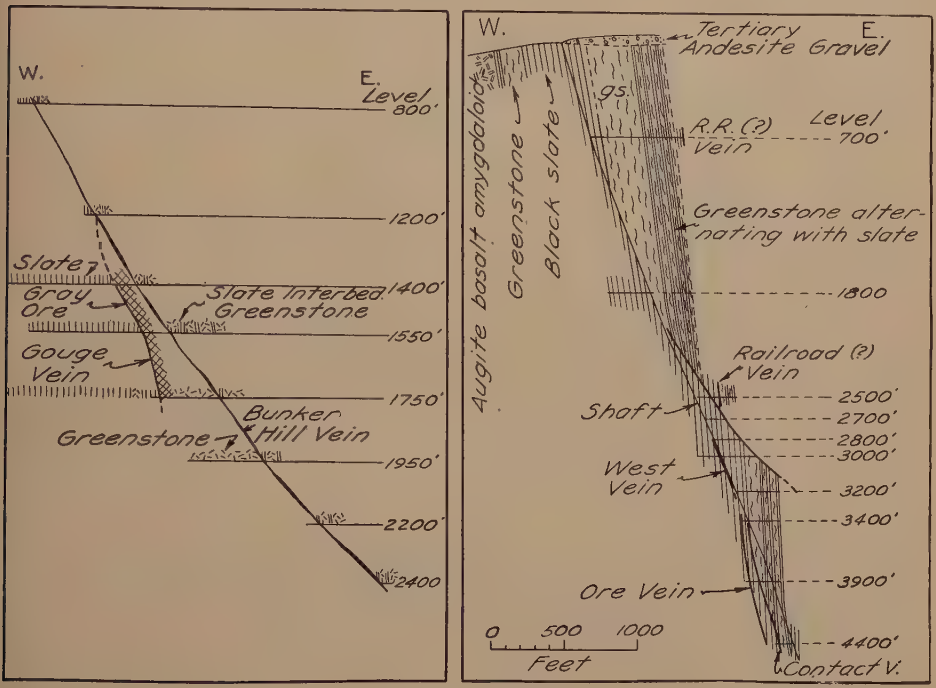
### *Mother Lode of California*

The Mother Lode[11a,34,57,68] extends along the west foothills of the Sierra Nevada for about 120 miles, striking northwest, its north end lying about 15 miles south of Grass Valley. It is a strip about a mile wide, with closely spaced, nearly parallel veins. The country rock includes slates, schists, and greenstones, which are intruded by serpentine and granodiorite. The bedded rocks dip steeply, generally to the northeast, and are cut by veins that cross them on strike and dip at small angles, generally to the northeast. The Mother Lode has produced nearly $300,000,000 gold.

The gold quartz veins fill fissures and along some of them is mineralized country rock. The latter includes the "gray ore," in part replacement of greenstone. Many of the veins fill reverse faults, the throw of which is about 700 ft. or less. Some of the faulting followed the deposition of some of the ore, the processes overlapping, as is shown by broken quartz cemented by ore. The fissures at many places cross the contacts of greenstone and slate. The greenstone is more resistant to fracturing than the weaker slate, and the fissures generally become flatter as they pass downward from slate into greenstones. They tend to remain in the slate because it is more easily fractured and to cross the greenstone more nearly directly because the greenstone is more resistant (Fig. 12) to fracture. Thus energy is conserved.

As at Alleghany, at many places the flat sections of the veins are wider than steeper ones (Fig. 12), although that is not invariably true. Some of the narrower parts of the veins that lie in slate are rich and some people believe that the carbonaceous slate exerted a favorable precipitating action on the ore-bearing solutions.

The flat parts of the veins in greenstone, although wider (Fig. 12), commonly are poorer than other parts in slate. Knopf believes that the veins were deposited by rising hot waters, which carried the metals from deep magmatic sources and dissolved silica from the country rock. He notes that near the veins the wall rocks have lost great quantities of silica. The greenstone, which at places is about half $SiO_2$, near the veins is converted to ankerite, at places almost completely. The wall rock was a source of silica that could easily have supplied part or all of the quartz and lowered the proportion of gold of veins in the greenstone. The flatter part of the vein is widest.



a b

Fig. 13.—Cross sections of mines of Mother Lode, California.

*a.* Geological cross section of Bunker Hill mine, 800 ft. north of shaft (after Knopf, U. S. Geol. Survey).

*b.* Geological cross section through shaft of Central Eureka mine (after Knopf, U. S. Geol. Survey).

The Original Amador vein is followed by the shaft (Fig. 12*b*), first in slate, then at the contact and below that in greenstone. It is widest in greenstone, but low grade.

The Keystone mine (Fig. 12*a*) between 1852 and 1920 produced $17,000,000, of which 15,000,000 came from veins in slate, chiefly from the West vein, and about 2,000,000 from the Contact vein on a reverse fault, chiefly where the footwall was slate and the hanging wall greenstone. The great quartz body of the Contact vein enclosed in greenstone is generally too poor to work (R. D. Downs, quoted by Knopf, p. 58 of ref. 34).

In the Central Eureka mine (Fig. 13*b*), the vein was barren to a depth of 1100 ft., where the top of an ore shoot was found (p. 60 of ref. 34). This extended down to about the 1900-ft. level, where it became leaner, and at 3000 ft. it was unworkable. The top of the ore shoot is at a place where greenstone is the hanging wall. Another vein, called the West vein, is developed from depths of 2800 to 3200 ft. From 3400 to 4400 ft., the Ore vein was worked.

The Argonaut mine, one mile northwest of Jackson, adjoins the Kennedy mine on the south. It is one of the deepest and most productive mines on the Mother Lode. The vein follows a reverse fault and cuts through greenstone and black slate. From the surface down the dip for about 700 ft. the vein is in greenstone and dips at a low angle (Fig. 14*a*). It becomes steeper in the slate and in general where one wall is slate and the other greenstone the dip is commonly intermediate; that is, it is steeper than where both walls are greenstone and lower than where both walls are slate.

The Moore vein is wider in amphibole schist than at the contact of the schist and underlying Calaveras black slate (Fig. 14*b*).

The Bull vein, Melones mine, Carson Hill, carries much quartz, not commercial ore, but has a workable body in the footwall and another in the hanging wall. The hanging-wall ore shoot lies in a trough made by a sharp bend of the Bull vein, and it is followed from the surface to the 4375-ft. level. It is crossed by five veins called the "Flat" veins, which have been worked, some yielding considerable ore. The "Flat" veins occupy reverse faults and offset the Bull vein and its valuable hanging-wall ore shoot. They dip from 20° to 35° north to northeast and vary from a few inches to 20 ft. or more in width[11b] (Knopf[34] gives a stereogram drawing by Thomas B. Nolan).

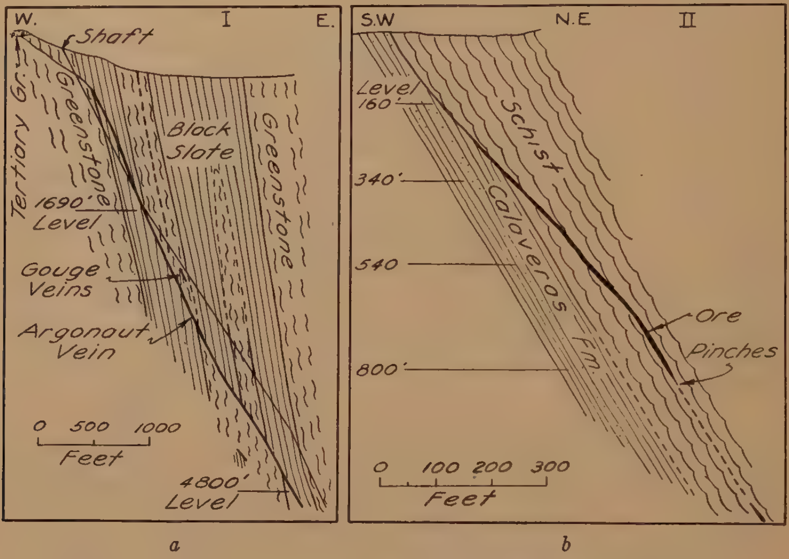


FIG. 14.—CROSS SECTIONS OF MINES OF MOTHER LODE, CALIFORNIA.
*a.* Geological cross section through Argonaut shaft (after Knopf, U. S. Geol. Survey).
*b.* Geological cross section through Moore mine on line parallel to and 60 ft. south of 640-ft. west cross cut (after Knopf, U. S. Geol. Survey).

### *Coeur d'Alene, Idaho*

The Coeur d'Alene[26,58,63,69] district, about 75 miles east of Spokane, Wash., has produced more than 800 million dollars in lead, silver, zinc, copper, gold and antimony. It is an area of pre-Cambrian quartzites, slate, and impure limestone highly folded, faulted, and intruded by granitic stocks and masses, probably in Cretaceous time.

North of Wallace and south of Kellogg, the Osburn fault strikes a few degrees north of west. It is a normal fault, dips south about 55° and has a great lateral throw. The south or hanging wall of the fault probably moved about 12 miles to the west.

The district contains two important mineralized centers, one north of Wallace and another near Wardner, 11 miles west of Wallace. The chief mineral deposits northeast of Wallace are great fissure veins and shear zones that strike nearly parallel to the Osburn fault. They cross the beds and strike nearly at right angles to the line of granitic intrusions that probably are the upper parts of a granitic batholith.

Since many of the deposits are veins in faults that are nearly parallel to the Osburn fault, it appears probable that the Osburn fault is in part older than the ore deposits, although some of the movement along the Osburn fault may be later than the ores. If a tracing of the map is cut along the line of strike of the Osburn fault and the south segment moved along the fault 12 miles to the east, or the amount of the horizontal displacement of the south side of the fault, the rocks would lie about as they did before they were faulted by the Osburn fault. The Wardner district would then lie at the south end of the belt of granitic intrusions and the zoning of metals with the south fringe of antimony and the east one of silver ores would be in the normal order. The suggestion is interesting, but, of course, the relation is not proved.

According to Umpleby and Jones (p. 142 of ref. 69) the mineralization is chiefly in and along the faults of reverse displacement. They say, as do others reporting on the district, that these faults have less gouge than the faults of normal displacement. This is explained by Umpleby and Jones as probably due to upward thrusts of the hanging walls of the reverse faults by movements of an underlying magma. The normal faults, having heavier gouge, were less favorable to movements of mineralizing solutions than reverse faults.

Very few cross sections of the larger veins of the district are available and there are few data that show that the flattest parts of veins are also widest. The Wardner center, south of the Osborn fault, is one of the most productive parts of the Coeur d'Alene district. The Bunker Hill and Sullivan mine has opened large deposits of lead-silver and zinc ores. These are sideritic replacements of pre-Cambrian quartzite and lie in a zone of complicated faulting and fissuring, which, according to Umpleby and Jones, are probably in part due to readjustments in the hanging wall of the Osburn fault. The chief deposits lie in the hanging wall of the Sullivan or Footwall fissure, which strikes northwest, dips about 35° to 45° southwest and is a normal fault marked by heavy gouge. Great ore bodies are found in the fissured faulted and crushed body lying above and within a few hundred feet of this fissure. The principal deposit is the March ore shoot in a zone of strong complicated faulting that is followed from the surface downward for more than 4000 feet.

The Sierra Nevada mine is about one mile west of Wardner. The Sierra Nevada vein joins the great Footwall or Sullivan fissure from the southwest (p. 160, ref. 58). It strikes east to northeast and dips from about 15 to 40 to the south. The widest part of the vein is where there is a low dip. This feature, however, may not be due to the flattening of a vein in a thrust fault. J. R. Finlay states that the vein where it was richest arched upward, like a low anticline plunging to the south.[22] Such upward arches of veins are commonly the loci of ore shoots.

In the Interstate mine, in an area where in general the sedimentary beds dip steeply, there is, on a structural terrace, a zone of flat-lying beds somewhat fractured, in which the mineralization occurs.[46,47]

### *Aspen, Colorado*

The Aspen[3,12,35,66,70] district, Pitkin County, Colorado, about 32 miles west of Leadville, is an area of tilted and faulted Paleozoic sedimentary rocks that were intruded by sheets and dikes of diorite porphyry, alaskite porphyry, and aplite porphyry. About five miles south of Aspen there are two stocks of granodiorite porphyry, which Knopf suggests may be connected with an underlying mass of granodiorite.

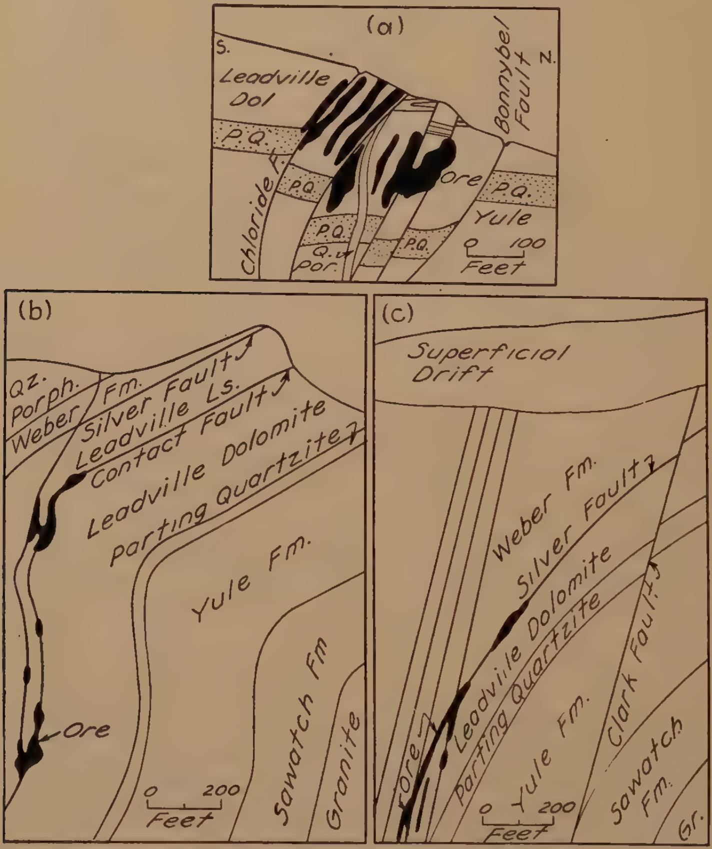


FIG. 15.—GEOLOGIC CROSS SECTIONS AT ASPEN, COLORADO.
(All from J. E. Spurr, U. S. Geol. Survey.)

*a*. Bonny Bell mine, N–S section nearly parallel to general strike of beds. In area of reverse faulting near Chloride fault, limestone is sliced and ore bodies (black) are in, near, or parallel to faults in area of lower dips. Other ore bodies lie near steeper segments of normal faults.

*b*. Geologic cross section, Durant and Aspen mines. Black is ore.

*c*. Geologic cross section, Park Regent mine. Black is ore.

The Aspen district has produced a little more than $100,000,000 silver-lead ore. Most of it has come from mesothermal replacements of limestone closely related to two faults that lie nearly parallel to the beds (Fig. 15). These are the Silver fault, between the Leadville limestone and the overlying Weber (?) argillaceous formation, and the Contact fault, which separates the dolomite member of the Leadville formation from the overlying Blue limestone.

Many of the larger deposits were found where cross faults cut the Silver or Contact faults. In some of the deposits (Fig. 15, *b* and *c*) the major mineralization lies on the steeper parts of the faults. The Bonnybelle mine is an area with certain reverse faults (Fig. 15*a*). Some of the deposits lie in or are nearly parallel to reverse faults and some of them lie in sliced blocks and brecciated zones nearly parallel to the segments of the faults that have the lowest dips. Other deposits on the same section lie on or near the steep segments of normal faults (Fig. 15*a*).

### *Montezuma, Colorado*

The Montezuma[42,50] district, Colorado, is in the Front Range about 50 miles west of Denver. It is an area of pre-Cambrian schists and gneisses intruded by pre-Cambrian granite. The pre-Cambrian rocks were eroded and covered by Cretaceous sediments. In early Tertiary time, stocks and dikes of quartz monzonite and other dikes were intruded. Fissure veins are found in all of these rocks. The Silver King (mesothermal) vein, about ½ mile south of Montezuma, occupies a normal fault that strikes about N.60°E. or more and dips 65° to 88° south. Proceeding westward, with the forward-moving block on the right, a swing of the strike occurs about 17° to the left, attended by an increase in width of vein to a workable deposit (plate 37, p. 102, ref. 42). The Bell vein (Fig. 16) is about one mile south of Montezuma. It is in a reverse fault with ore shoots in its flat segments.

### *Illinois-Kentucky Fluorite District*

In southern Illinois and western Kentucky[2,4,13,73] there is probably the greatest fluorite district known. It is an area of Mississippian limestones, sandstones, and shales, folded, faulted, and intruded by many basic dikes. The deposits are veins, filling and replacing limestone along fault fissures, and replacements of limestone beds below shale. The fluorite is associated with calcite, barite and quartz, and some galena, sphalerite, pyrite and chalcopyrite.

The largest deposit of the district is the Rosiclare vein (Fig. 17), which occupies a fault fissure, strikes east of north and is nearly vertical from the surface to a depth of about 350 ft., where it dips northwest about 83°. The fault is normal and the throw on the northwest side is about 200 ft. The vein is a filling deposit with breccia.

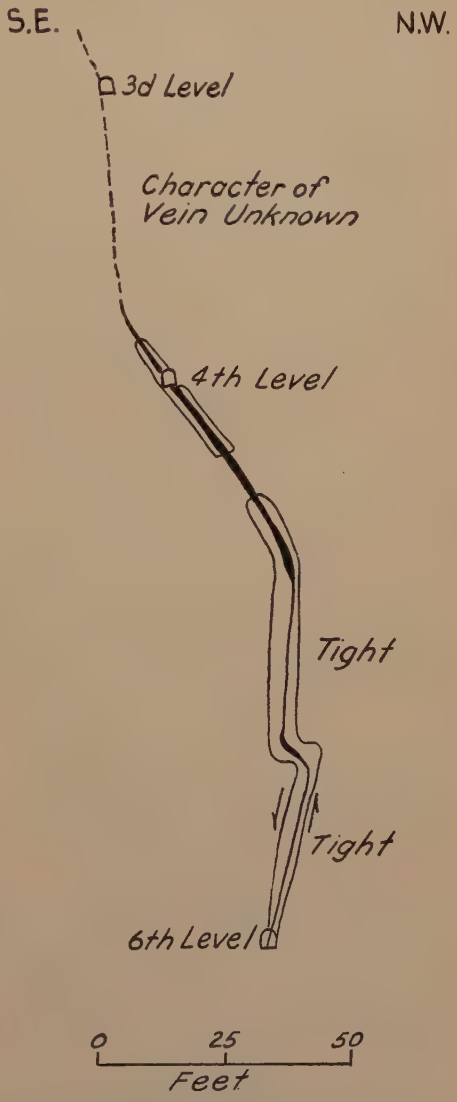


Fig. 16.—Section along raise in Bell mine, level 6, one mile south of Montezuma, Colorado.

Vein occupies reverse fault and its flatter segments are wider than steeper ones (after T. S. Lovering, U. S. Geol. Survey).

It is probably mesothermal, although the minerals are chiefly persistent ones and not diagnostic. The width of the vein varies from 3 ft. to about 25 ft., the wider parts being above the depth of 350 ft. where the dip is steepest. It becomes very much narrower at lower levels, where its dip is lower (plate III and p. 70, ref. 4).

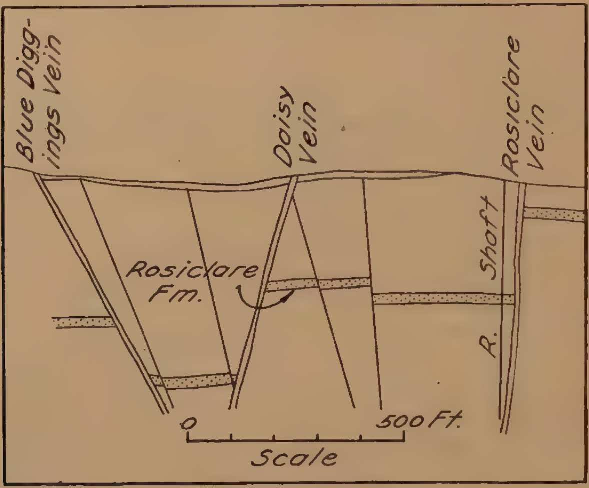


FIG. 17.—DIAGRAMMATIC CROSS SECTION SHOWING ROSICLARE DAISY AND BLUE DIGGINGS VEINS, HARDIN COUNTY, ILLINOIS.
(After sections by Weller, Currier, Bastin and others.)
All veins are in normal faults, widths are generalized. Dip of Rosiclare vein decreases with depth and its width also decreases.

### HYPOTHERMAL DEPOSITS

Hypothermal deposits in general are formed in the deep zone, commonly 2 miles or more below the prevailing surface at the time of their deposition. The deposits are formed at high temperatures and pressures. Veins in faults of which the displacement is known are very rare, although they are found even in normal faults in certain tin-bearing districts. The great low-angle reverse faults that outline many mountain chains are not known to carry valuable mineral deposits. Probably they carry too much gouge to permit the passage of metalliferous solutions.

### *Rossland, B. C.*

The Rossland[15] district, British Columbia, about 5 miles southwest of Trail, has produced about $85,000,000 in gold, copper, and silver ore. The country is an area of slates (Carboniferous) which were deformed near the close of the Paleozoic time and were intruded by augite porphyrite probably in late Triassic time. Near the close of Jurassic time these beds and igneous rocks were greatly deformed and intruded by the Trail batholith, and there were also extrusions of lavas and tuffs.

The first period of mineralization took place at this time, shear zones being formed along fissures largely at contacts of the Trail batholith and roof rocks and in the roof of the batholith. These fissures and shear zones were mineralized about the close of Jurassic time. During Cretaceous time the roof of the batholith was largely eroded and in Laramie time the veins were faulted. Eocene erosion and sedimentation followed and in Oligocene (?) time there was deformation, intrusion of porphyritic monzonite stocks and of lamprophyre dikes. In Miocene time the Coryell batholith and other intrusions were emplaced. There was a second period of mineralization in the Miocene time, attending intrusion of stocks and dikes and still later many lamprophyre dikes were intruded. There are many faults in the Rossland district, but these in general have small displacements.

The chief deposits are replacement veins in augite porphyrite and shear zones generally along contacts. The deposit in 1687 raise, LeRoy mine, has a hanging wall of granodiorite and a footwall of augite porphyrite. It is a zigzag vein and its steeper segments are widest, which suggests that it is a normal fault (p. 49, ref. 15).

### *Cornwall*

The tin-copper district of Cornwall and Devon,[14,28,45,61] England, from 1201 to 1905 produced about 1,467,890 tons of metallic tin and 903,350 tons of copper. It is one of the world's greatest mining districts, but for the past 20 years most of the mines have been closed.

The country is an area of folded Paleozoic sedimentary rocks, chiefly slates and sandstones, but with some igneous rocks that were intruded near the close of Paleozoic time by five great granitic bodies and by 10 or more small stocks. The ore deposits are fissure veins from 3 to 40 ft. wide, which fill openings and replace or impregnate the country rocks along the openings. They are found in granitic rocks and in the invaded rocks and nearly all of the important ones are at the margins of the larger granitic bodies, and particularly in or near the smaller granitic stocks that are believed to be cupolas, extending upward from one or more great batholiths.

Some of the veins are followed 3½ miles on strike and 3000 ft. in depth. Many of the veins are fault fissures. Even normal faults are mineralized and the lodes of the area in which tin and copper occur—that is, in the part of the district containing the most deeply seated types of deposits—important veins fill faults

Evidences that the veins occupy faults as stated by Hill and McAlister (pp. 144–150, ref. 28) are: (1) dislocation of features recognized on both sides of the vein; (2) forms of the fissures and failure of opposite walls to fit; and (3) brecciation of wall rocks. The displacements along faults are generally small, although one of 120 ft. is recorded. There is little detail as to the effect of change of dip on ore shoots, but it is stated that the veins are widest near places where their strike changes.

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# The Search for Concealed Deposits—a Reorientation of Philosophy

By Samuel G. Lasky,* Member AIME

(New York Meeting, March 1947)

> *What the evidence prevails upon the mind to believe,*
> *depends upon the mind as well as upon the evidence.*
> M. L.

In a recent discussion of the mineral resource position of the United States, published in the September 1946 issue of Mining and Metallurgy, I expressed my conviction that ore deposits of major magnitude remain to be found in this country and that they can be found without undue cost or risk. I stated with equally strong conviction, however, that if the search for such deposits is to be notably successful there must be, among other things, a reorientation of philosophy and mental habits on the part of many who would be connected with the search. I would now like to enlarge upon that statement.

For such a purpose, we may consider that the search for concealed deposits involves four steps: (1) the selection of areas for further study; (2) the geologic study of those areas in order to verify their general favorableness and in order to choose specific patches worthy of being tested; (3) the decision whether exploration of the patches chosen should actually be undertaken; and (4) the exploration itself. The first two steps and the last rest upon the judgment and skill of geologists. The third step—the decision whether exploration should be undertaken—rests upon management. In my opinion, the philosophies or principles that have guided these steps in the past are now inadequate.

## Philosophy of Appraisal

If concealed deposits are to be looked for, how are the places of their possible concealment to be recognized? Every known mineralized area still in the prospect stage has already been repeatedly examined and condemned. As for those deposits concealed under later rocks, there may be not even the clue of a near-by outcrop for engineer or geologist to consider. How then, are we to know where to look?

### *Geologic Probability*

In the early days of our mining industry, ore deposits were found mainly by "following the ore." Later, as such opportunities decreased, and as geologic concepts became accepted, deposits were found more and more through use of the notion of geologic probability. Many instances can be cited wherein a courageous operator ignored the implications of the unsuccessful efforts of predecessors and, solely by "following the ore," converted a discredited prospect into a good mine. Similar experience doubtless will occur in the future, but "following the ore" will no longer in itself find new deposits in the amount needed. Nor can enough new ore be expected from further use of the notion of geologic probability, for, as mentioned above, our known prospects have already been ex-

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Published by permission of the Director, U. S. Geological Survey. Manuscript received at the office of the Institute Sept. 5, 1946. Issued as TP 2146 in Mining Technology, May 1947.

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amined time and again and as often condemned on the basis of this notion. It can even be said that of those prospects that eventually were developed into mines, many became mines more by virtue of repeated examination than through prompt and true appraisal of the geologic probabilities. Every mining man is acquainted with the usual history in such cases: the prospect is examined by a scout or engineer who, on the basis of his analysis of the geologic probabilities, recommends specific work to be done. The work is done but is unsuccessful. In due course, another scout examines the property. Having a copy of the first man's report—or deducing the other's reasoning from the work done—and being further guided by the results of the earlier work, this second scout develops a second theory of ore occurrence and recommends further work based upon it. These steps may be repeated still a third or fourth time, but eventually an ore body is disclosed as the exploration openings are advanced bit by bit.

Whether it be done consciously or unconsciously, the usual procedure is to look for evidence that ore may be present, the strength of the evidence being appraised on the basis of geologic probability as determined by accepted geologic concepts. Not only is this the usual way of viewing the evidence, but it is almost a fetish. Many persons whose duties include review of another's recommendations have had countless experiences in trying to induce him to lay out a plan of exploration that would test several theories of ore control at the same time, instead of only the single theory representing his idea of the greatest geologic probability. The fallacy of the doctrine of geologic probability is that the final inference as to the presence or absence of ore, or its most likely position, is built upon a succession of previous inferences or assumptions, even though unconsciously made, concerning a variety of pertinent but often poorly understood geologic factors. Thus the so-called "probability" may in truth be extremely remote. We need only list a few of these factors to see how true that statement is: (1) the mode of emplacement of the igneous rocks with which the ores are spatially associated; (2) the supposed genetic relation between igneous rocks and ore; (3) rock alteration, and the relation of alteration to ore; (4) the nature of ore-forming solutions and the mechanics of their accumulation; (5) geologic structure in general and local preparation of the ground in particular; and (6) various other factors that constitute "ore controls" from district to district or from mine to mine. At any rate, geologists who see their interpretation of what lies beyond their vision put to the test, sooner or later decide that nature delights in doing what the geologist calls improbable. Joralemon[1] expressed this idea in the question, "Is there something in the nature of ore bodies that will always defy any laws we may deduce, like 'sports' in the realm of natural history?"

Although the examples described by Joralemon are comforting, the unexpected in the search for ore more often than not is distressing. It is this unhappy fact that is responsible for the present national concern.

I do not wish to deny the past utility of the usual procedure in appraisal, nor do I wish to recommend that it be discarded entirely in the future; for, without the geologist's best judgment of what is most probable, exploration would be haphazard, or capricious, or based on strictly geometrical concepts. The use of the doctrine of geologic probability is natural, and under past conditions of abundant opportunities it represented the sensible approach. It is still the best approach in search for extensions of known shoots or for new shoots in active mines where the geological pattern is well developed. But the past habits of thought are no longer completely appropriate. The creation of a productive mining district involves so many geologic factors about which geologic thought is still highly theoretical that if we continue to use these habits in a search for concealed deposits we shall be placing a high premium indeed on our ability to reason out the greatest geologic probability. Since we have already appraised all known occurrences on that basis, we would be handicapping ourselves almost into helplessness, or into reliance on the theory that the necessary new ores will be found in the natural course of events—that is, that "the Lord will provide."

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[1] References are at the end of the paper.

### *Geologic Permissibility*

I am by no means "selling geology short." But if I am correct in my belief that procedures by which ore deposits have ordinarily been sought are no longer adequate, what course is to be followed?

I would recommend serious consideration of any mineral occurrence where the geology would permit, first, enough space or room for a mine or district and, second, ore in the necessary quantities close enough to the surface to be exploited. Instead of looking for information indicating that ore is likely to be present, we ought deliberately to attempt to balance and fit together the observed facts so as to discern whether ore could be present. In other words, the geology need be only permissive instead of definitely indicative. Accordingly, I propose the concept of geologic permissibility to replace the wholehearted use of the concept of geologic probability. Instead of confining himself to the question, "Should ore be here?" the geologist should also ask himself, "Could ore be here?" Clearly, this proposal does not completely reject the theory of geologic probability, for in some circumstances that theory would still have to be used as an aid in deciding which of several permissible localities to study or explore first.

Specifically, this recommendation mentions only "mineral occurrence" but I intend it to apply not only to known mineral occurrences but also to areas that may contain deposits blanketed by younger rocks. Such areas fall into four categories: (1) those that lap onto known mineralized ground, whether that ground is now productive, was once productive but is no longer so, or never was productive; (2) those that lie between known mining districts, such as the valley areas between mining districts in the desert ranges of the Southwest; (3) those that are near a known mineralized area, perhaps a fraction of a mile to several miles away, their favorableness being suggested only by projection of geologic trends; and (4) those that are so far away from a known mineral occurrence, or from other geologic features with which ore is sometimes associated, that geologic trends cannot be projected to them on any logical basis.

The fourth category may represent a hopeless situation, perhaps even with the development of extremely sensitive geophysical methods, unless the cover is sufficiently shallow to permit geophysical observation through it—a point I hope to discuss later. The other three, together with areas known to be mineralized but as yet not greatly productive, constitute our main hope of replenishing the nation's reserves by new discoveries; and the philosophy of appraisal recommended above for the known mineralized areas applies to these three as well. The only difference is that the appraiser, increasingly as he moves farther from the known to the unknown, must be more consciously optimistic, more imaginative, and more willing to "stick his neck out" to a degree ordinarily distasteful.

Careful geologic mapping of any mining area, whether a few square miles or a few hundred square miles, by a geologist alert to the purpose of his mapping, almost invariably leads to recognition of hopeful prospecting sites. These sites may be at known prospects that previously, when examined individually or independently, had been condemned; or they may be at places theretofore entirely unsuspected of containing ore.

Once it is concluded that the geology at a given locality may permit a mine or district to be present, the critical spot, both in the geological reasoning and in space, can then be tested by exploration. As an example, let us assume that two faults, known to be ore channelways, extend under an area of younger rocks and that this area is considered favorable because it has been reasoned that the faults join and that, as a result, they bring a favorable ore horizon within reasonable distance of the surface. The critical point in the geological reasoning is that the faults join, and the critical point in space is the position of the favorable horizon. If the test is disappointing in critical respects—if the faults do not join, or if the favorable horizon is too deep—it is a simple matter to drop the project, with but little loss. If the test proves the geologic reasoning sound or, fortunately, actually discloses ore, it is an equally simple matter to test the then critical point, again both in the reasoning and in space; and the process can be carried on until it becomes clear whether or not a mining enterprise can be established.

This idea of geologic permissibility is not completely new, for the oil industry acts upon it as a matter of habit, requiring only permissive structure and some indication of source rocks. The exploration departments of at least two large mining companies act upon it from time to time, but it is by no means common in the mining industry. Even less is it as much a matter of habitual policy as it should be if the nation is to discover the additional ores needed to avoid the impoverishment that some believe is already here.

### *Nature of Geologic Maps*

Since geologic maps commonly constitute an essential part of the information on which favorable areas are chosen for further study, some analysis of the nature of geologic maps is pertinent here. Too often the fact seems to be overlooked that a geologic map is as much a product of geologic art as of geologic science or of simple ability—that it has subjective as well as objective elements. Two lathe operators, equally skilled and working from the same pattern, will produce identical objects; but no two geologists will produce identical maps of the same area. A geologic map bespeaks not only the skill of the maker but also his opportunities, his specialty, his temperament, and his purpose. Different geologists will use different scales and will show different amounts of detail, different degrees of accuracy, and, indeed, different relative amounts of fact and inference. Theoretically, the amount of detail shown on a map is controlled by scale and perhaps by some consideration of the point of diminishing returns; but normally it is the geologist himself who chooses the scale and who decides when the point of diminishing returns has been reached. His basic considerations are the texture of the features to be mapped and his interpretation of the purpose of the work.

The accuracy of a map and the relative amounts of fact and inference shown depend on skill and temperament. Some geologists like to map infinite detail; others like to generalize. Some pursue interesting items as long as something new continues to be learned; others lose interest—or at least stop—when the time and effort begin to outweigh what they expect to get out of it, that is, when they reach the point of diminishing returns. Some believe that good geologic mapping is mainly leg work and observation, albeit guided by inspiration, considering it a matter of practical consequence if their deductions prove wrong. Others believe mapping to be only an exercise in sketching and a good-humored tussle with nature, in which the main requirements are skill in deduction and the ability to win or lose gracefully. Some geologists make a geologic map mainly to illustrate their interpretation of the geology; others make a map initially as a device to study the geology. Some maps are made with no specific consideration in mind as to practical use; others are initially inspired by some practical purpose.

The net result of this diversity of temperament and purpose—not to mention specialization and opportunity—is that different geologic maps of the same area may be as different from one another as portraits of the same individual by different artists.

In many respects these comments about geologic maps are self-evident, but I think they are pertinent because I know of areas that have been chosen for commercial study mainly on the basis of information compiled from old maps, without any special discrimination, as if those maps were certain to contain "the truth, the whole truth, and nothing but the truth." The primary consideration in accepting the maps without question seems to have been that the authors of the maps had good reputations; but if the difference between geologic mapping as an art and geologic mapping as a science be kept in mind, the imperfections of a map can be heeded without the ability of its author being impugned.

### PHILOSOPHY OF GEOLOGIC MAPPING

I have contended earlier in this paper that the philosophy of appraisal used in the past, in deciding where to look for new deposits, has become obsolete. The mental attitude assumed by the geologist usually called upon to do mapping in mining districts is likewise obsolete, in the sense that it is inadequate in the search for concealed deposits. A characterization of geologists in general made recently by H. R. Aldrich,[2] secretary of the Geological Society of America, may profitably be repeated here. "Geologists," he says, "are as skilled carpenters fashioning new structures from well-known building material, using the same old kit of tools and following dusty blueprints"; they continue to "map new areas by old methods, work up new collections by old techniques."

No simple mental portrait is evoked, of course, when one speaks of the field geologist. In fact, some geologists may be called upon to map areas that cannot possibly contain ore deposits. My comments are intended to apply to the geologist who does the mapping of large areas as a necessary preliminary to the search for concealed deposits. Like all other geologists, geologists who do work related to ore geology derive their inspirations and attachments from many sources. Some are on company staffs, doing diverse kinds of work; some are full-time consultants; some are teachers who accept consulting commissions from time to time; and some are on the staffs of State and Federal bureaus —but I think it fair to say that field geologists are in a sense all cast in the same mold. On the average, the field geologist in approaching his work has been instinctively motivated more by scientific interest than by the urge to find ore. He approaches the job with the idea first of making a geologic map and then considering what the information thus acquired means in terms of the search for ore. The making of the geologic map is his basic interest, and its application to the search for ore seems often to be in the nature of a belated recognition of duty. As phrased by Augustus

Locke,[3] "The real motive of the geologist is not the commercial urge, but his own interest in achievement along his own peculiar lines." . . . "Detailed critical study in mining districts," he continues, "is extremely uneven and spotty. Here and there it indisputably controls [in ore discovery] . . . , but in most districts it is haphazard and amateurish. The most consistent efforts in the direction of a science of ore occurrence have been made by men interested in scientific progress for itself. Altogether the course of ore geology has been governed by scientific fervor and not by the demands of ore discovery."

### *Techniques and Habits of Thought*

There are, of course, reasons other than the mental attitude and temperament of geologists why this indictment can be made. One is the inertia toward change in human affairs. Techniques and directions of thought that are established to meet a specific need or purpose imperceptibly tend to become habitual, even though the conditions that led to their creation change or disappear. Not so many years ago, district and regional geologic mapping was done solely by federal or state agencies or by graduate students working under the auspices of the universities. The purpose of the student was to make a geologic map and to write some sort of accompanying text in order to prove to the faculty that he was qualified to receive the degree for which he was a candidate. The primary purpose of the state and federal surveys was, as phrased by W. H. Emmons,[4] of the Geological Survey, "to endeavor to determine the general laws which govern the formation of ore deposits." The practical application, "while a most desirable object, was considered of relatively secondary importance and one to the accomplishment of which no jot of scientific accuracy should be sacrificed."

In addition, the state and federal surveys were in many respects training schools for geologists who transferred their activities to industry as the practical usefulness of geology became more and more established; and these men carried with them the habits of thought and practice they had been taught.

There are signs that this propensity of the field geologist more toward scientific achievement than toward practical application is declining in ore geology, particularly among geologists attached to company staffs; but the change is not a conscious one nor has it yet gone very far. In the Geological Survey, G. F. Loughlin, as head of the Section Metalliferous Deposits, began the practice of pointing out, in the Survey's reports on mining districts, the practical application of the geologic studies; and a still further practical step was taken by his successor, D. F. Hewett, even prior to the war work of 1939–1945, in the sampling of some deposits for an independent appraisal of reserves. Even so, many authors have treated these matters as annoying chores that had to be done because they were told to do them. One of the outstanding economic geologists of the country, a member of a public survey, once told me that he was interested in mines only because they gave him greater access to the third dimension than did topography alone.

### *False Distinctions*

In a sense, there exists what I consider a false distinction between geologic research and applied geology, for the finding of ore calls for the highest scientific skill and aptitude a geologist possesses. In the field of ore geology the commercial geologist who scorns geological research, and the research geologist who scorns or gives no thought to the possible practical application of his discoveries, are both defeating their purposes. Indeed, even if their purposes are only to do work they enjoy, the mental exhilaration accompanying the search for and discovery of scientific truth can be equaled by that which comes from the successful application of the discoveries.

### *Recommended Practices*

There is a principle in statistical research that, I believe, indicates why past practices and habits of thought are inadequate in the search for concealed deposits; i.e., that data used in a problem should be collected for the purpose of the problem, and that conclusions based on data collected for other purposes are almost doomed to error or inadequacy. When the practical applications of a geologic study are considered as an afterthought or as something for "the other fellow" to consider—if the geologic studies are motivated, even though unconsciously, mainly by an interest in scientific progress as an end in itself—the data are not collected for the specific purpose of the problem, the conclusions are less sound than they should be, the results less complete. The geologist could well ask himself at each new observation and as each new line is drawn upon the map, "What may be the meaning of this in terms of the search for ore?" I have been asked if this suggestion is not hyperbole. On the contrary, I mean it almost literally, for even something no more complicated than a change in strike and dip may be evidence of significant structure. I believe that under such practice the geologist will observe not only everything he would have observed otherwise but he will observe much more of significant value as well; and the results of his study will not only be more satisfactory in a practical sense but also will constitute a better scientific product.

I would urge, too, that this practice be coupled with two others—conscious oriented speculation and conscious rigid appraisal of the significance and validity of basic data. Geologists are taught caution of thought—such caution theoretically being incompatible with speculation—but there is a difference between, on the one hand, wild unconscious speculation that through some alchemy of enthusiasm becomes transmuted into conviction, and, on the other hand, conscious speculation and imagination oriented toward the purpose of the job and arising out of a mental alertness that is itself nourished by that purpose.

True, danger always lurks in speculation, but the conscious speculation of which I speak can be steadied by a rigid appraisal of the validity of evidence, and likewise can receive added value from it. Evidence may be direct or indirect, proving a point forthwith or only through related circumstances. It may be exclusive and mandatory, excluding all but one interpretation and demanding that one; it may be permissive only, permitting an interpretation but not demanding it; or it may be no more than suggestive. Likewise, evidence is acquired under a variety of conditions that influence both its validity and its significance. Not the least of these are styles of thought and current theories, the validity of which should be scrutinized no less than the validity of the evidence itself. Theories that depend for their acceptance, in the words of Sir Arthur Holmes, "largely upon the weight of external authority," are particularly open to question.

I fully realize that the successful application of my recommendations about geologic mapping depends on temperament and that the practices recommended may therefore be charged with impracticality. They would, indeed, be impractical if they were intended to apply to all geologists; but only those geologists whose interests lie with the development of scientific geologic guides and tools for the discovery of ore deposits, and with their practical application, should be entrusted with the work of searching for concealed deposits. Such men, I am sure, have the temperament to apply these recommendations successfully. There is an abundance of essential work in other fields for geologists with other temperaments. But some geologists who hitherto have devoted themselves to geologic research for its own sake may come to feel that a job that proves a scientific truth, solely for the sake of proving it, and then stops, is but half done; and, as happened during the war, these geologists may find themselves, by a reorientation of effort, capable of and interested in putting their research to practical use.

### Management

This leaves for discussion my early reference to the role of management, which must decide whether or not areas or patches of ground certified by the geologist are to be explored. I shall not presume to enter this subject far, because it has ramifications in fields that are beyond both my experience and the purpose of this paper. There are, however, some geologic aspects of it that may be mentioned.

Strictly speaking, ore is never "found" by means of geology. The most that a geologist can do toward "finding" ore is to indicate the place where the ore would be if nature were kind enough to put it there. Thereafter, someone must make a hole, of some kind, for visual evidence that ore is present.

Neither can a geologist prophesy the actual size and shape of a possible or probable ore body or its actual tenor. He can, however, by analogy with producing areas or mines where the geology is significantly similar, make a good estimate of the over-all magnitude and general grade. He can also express judgment on the possible size, shape, distribution, and attitude in space, of the shoots that may compose the whole, and thus arrive at a good estimate of the cost of exploration and subsequent development as compared with the possible and probable rewards. Not only can the geologist do these things, but he has a better judgment in such matters than anyone else; and it would further the likelihood of successful search if management accepts that judgment, remembering, however, that such expressions of judgment are opinion and not prediction.

Good business, of course, requires a satisfactory return on the investment and some assurance that such a return will be forthcoming. Whereas appraisal of the gamble is a function of the geologist, the decision as to what return and what assurance are satisfactory is a function of management, but it is my belief that in the coming days management will have to be more liberal in what it is willing to accept as satisfactory assurance—be willing to take a larger chance than in the past.

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New deposits are harder to find than they have been, but that does not mean that the difficulties are insurmountable nor even that they are unreasonably great. It does mean that more courage and vision, both intellectual and financial, must be shown. The rewards will be monetary return for the entrepreneur, intellectual achievement for the geologist, and satisfaction for both in that they have recognized a personal opportunity and have met a patriotic obligation.

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## DISCUSSION

T. M. Broderick*—When anyone, from the humblest prospector to the explorer employed by a large mining company, decides to dig or drill, in most cases he chooses a place to spend the money which to him has a better chance of success than some other place. We all recognize the possibility, even more, the likelihood that under the glacial drift, or the valley wash, or the later flows, ore bodies or even potential mining districts may lie concealed. But until methods come to light of which we do not yet know, there will probably not be enough exploration capital in the world to test out many such places where ore is "geologically permissible." Therefore, we must continue to spend our money in places which we think have better chances than other places.

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Some one must assume the responsibility of rating the different places to explore in the order of probability of successful outcome. If he is a geologist his rating is to a large extent one of "geologic probability." He bases his appraisal of localities upon the application of the existing knowledge of the geology of ore deposits. He knows that this knowledge is incomplete and fragmentary and that perhaps some years hence he would rate the probabilities of the same places in a different order. But after all he must make a decision right now and his choice is based upon "geologic probability" as he sees it today. (For the purposes of this discussion other factors which must be appraised, such as accessibility, political considerations, distance to water, and so on, are assumed to be constant.)

There are districts where the ore controls, accompanying rock alteration and other features upon which "geologic probability" is based, are more obscure than in other districts. In such cases mere "geologic permissibility" plays a greater part in guiding exploration. It requires greater fortitude on the part of both the geologist and the management to spend money where the best that can be said is that ore *could* be there rather than *should* be there.

At any rate, geologists will do well to not let the idea of exploring on the principle of "geologic permissibility" get out of hand. What a new lease on life the promoters with a smattering of geology would get. How the old choruses would ring out like, "Where it is there it is," "The land ain't good for anything else so it must have gold."

# Mineral and Metal Variations in the Veins of Fresnillo, Zacatecas, Mexico

By J. B. Stone,* Member A.I.M.E., and J. C. McCarthy†

(New York Meeting, February 1942)

At Fresnillo a series of veins that has yielded very large quantities of silver and other metals has been developed over a length of 6500 ft. and to a depth of over 3000 ft. In the course of this work striking changes in the mineralogy and metal content of the principal veins have been disclosed and are well illustrated by assay data and ore reserve calculations. This paper describes those changes and suggests their bearing on theories of ore deposition.

## Location

The Fresnillo mines and the adjacent city of Fresnillo are in the central part of the state of Zacatecas, 35 miles northwest of the capitol city of Zacatecas, once a great silver-mining center whose total production is said to place it fourth in the world. Another famous old silver district, Sombrerete, lies northwest of Fresnillo.

Fresnillo is connected by a 5-mile branch with Fresnillo station on the Mexican National Railways line from Juarez to Mexico City.

Lying near the southwestern edge of the central plateau of Mexico, Fresnillo is surrounded by a semi-arid rolling plain from which rise widely scattered low hills. One of these is Proaño hill, an inconspicuous elevation rising barely 300 ft. above the plain, but in and around Proaño hill are the Fresnillo mines, and it is marked now by a great gash cut by glory-hole mining.

## History

According to an account written in 1834 by Francisco García, then Governor of the state of Zacatecas, Fresnillo was discovered in 1554, and a presidio was established there in 1568 to protect travelers on the road between Zacatecas and Sombrerete.

Little is known of the earliest mining at Fresnillo, which seemingly began soon after its discovery, but by piecing together various accounts, it appears that mines there were working in 1717 and were notably active in 1751 under an administrator named Murguía, who left detailed notes describing them. In 1757, however, the mines became inactive because of difficulties with creditors and no doubt also because of increasing water troubles, and they remained abandoned until 1830.

In 1830 the mines at Fresnillo became the property of the State of Zacatecas because the interest of Governor García had been aroused by the long disregarded notes of Murguía.

Under García's energetic leadership the mines were reopened, and silver production began in 1832 in spite of serious difficulties, including abnormal rains and an epidemic of cholera. In 1835 English capital was sought to allow installation of steam-driven Cornish pumps at the two principal shafts and to build the great *hacienda de beneficio* (the present Proaño hacienda). The company formed in 1835 (Cia. Zacatecano-Mexicana) operated successfully until 1872, when its career was brought to an end by the political and economic disturbances

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Manuscript received at the office of the Institute Feb. 3, 1942. Issued as TP 1500 in Mining Technology September 1942.

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then affecting all of Mexico. By 1859 the mine workings had reached a depth of nearly 400 meters.[1]

From 1878 to 1903, activity at Fresnillo was confined to work above water level and on old dumps. In 1903 the Fresnillo Mining Co., an American company, erected a lixiviation plant for treatment of the patio tailings, and in 1911–12 it acquired the mines also and built a 500-ton cyanide mill for treating the large tonnage of oxidized and partially oxidized pyritic silver ore, which it had proved by sampling the old shallow workings.

All work ceased from 1913 to 1919 during the worst of the revolutionary period, but in 1919 the property of the Fresnillo Mining Co. was leased to the Mexican Corporation, an English company; a 2200-ton cyanide plant was erected and large-scale mining of the shallow silver ores began in 1921.[2,3] In 1929 the present company, The Fresnillo Company, was formed by a merging of the interests of the Fresnillo Mining Co. and the Mexican Corporation. Meanwhile complex sulphide ore containing lead, zinc, and copper had been discovered in 1925, and the first flotation unit for its treatment had been built.[4,5] It was found on development that the previous operators had encountered the complex sulphides in the western part of the mine but had avoided them as "rebellious" to amalgamation.

At the present time, mining of oxide ore is unimportant, and the main production is from sulphide veins. The deepest workings have now reached 965 m. (3165 ft.) below the mine datum, which is an adit level approximately 100 m. below the top of Proaño hill.

## Production

Like its early history, Fresnillo's metal production prior to 1919 is imperfectly known, as the minutely detailed records of the nineteenth century have by now largely disappeared.

The production before 1832 is entirely unknown but could not have been very large by modern standards. The production from 1832 to 1903 was calculated by Church[6] (1906) at a time when some of the old mine records survived, and the values given by him (U. S. currency) were as follows:

| | |
|---|---|
| 1832–1835 | $ 6,454,697 |
| 1835–1872 | 41,341,851 |
| 1879–1903 | 4,784,030 |
| Total | $52,580,578 |

This total may be taken as the equivalent of some 43,000,000 oz. of silver, as contrasted with Livingston's[3] statement (1932) that old dumps and tailings indicated a past production of 4,000,000 tons of ore, from which he estimated that some 80,000,000 oz. of silver had been recovered.

The amount of silver and gold produced from dumps and tailings in the period 1903–1913 is not available to us at present.

From 1921 to 1941, inclusive, the Mexican Corporation and The Fresnillo Company produced 14,218,962 tons of oxide ore (including some tailings) averaging 5.58 oz. of silver and 0.175 dwt. of gold per ton. From 1926 to 1941, inclusive, 4,940,833 tons of sulphide ore have been mined, averaging 12.33 oz. silver, 0.53 dwt. gold, 5.05 per cent lead, 5.60 per cent zinc, and 0.50 per cent copper.

The gross silver content of the ores mined since 1921 can be calculated as 140,254,970 oz. The Yearbooks of the American Bureau of Metal Statistics give the production of the Mexican Corporation and The Fresnillo Company from 1921 to 1940, inclusive, as 112,814,271 oz. of silver, but this includes a small proportion from ore from other districts. It is quite certain that the recovered silver content of the ores mined at Fresnillo since its discovery has been in excess of 150,000,000 oz., or 5143 tons.

## General Geology[7]

The stratigraphic section at Fresnillo is as follows:

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[1] References are at the end of the paper.

Tertiary (?) volcanics and sediments
  Basalt flows
  Rhyolite flows and interbedded tuffs—about 500 m.
  Tuffaceous sediments—over 200 m. thick
  Sedimentary breccia—maximum thickness at least 200 m.
Mesozoic (?) sediments—Proaño group
  1. Upper graywacke—more than 300 m. thick
  2. Carbonaceous and calcareous shale—about 180 m. thick
  3. Lower graywacke and clay shale—more than 500 m. thick

Graywackes, shales, and subordinate limestones of probable Mesozoic age, here called the Proaño group, are found from the top of Proaño hill to the deepest mine workings, a vertical distance of more than 1000 m. They can be divided into three lithologic units, as indicated above. The upper graywacke consists of graywacke, shale and limestone, with a thick bed of massive graywacke at its base. The shale unit has carbonaceous and calcareous shale above, grading down into shaly limestone. The lower graywacke and clay shale is less diverse lithologically than the upper graywacke. The clay shale might more properly be called argillite.

Probably at the end of the Mesozoic era the Proaño rocks at Proaño hill were folded into a large gentle anticline trending west-northwest. Incompetent shale and limestone beds show extreme minor crumpling in places in spite of the gentle dip of the larger structure. A land surface of considerable relief was eroded on the folded Proaño rocks, and on this surface was deposited an angular detritus derived from the graywackes and shales themselves. This sedimentary breccia (locally called "agglomerate") has a maximum thickness of more than 200 m. (Fig. 2).

The breccia is overlain by at least 200 m. of tuffaceous sediments, the youngest rocks of Proaño hill. Other hills near Fresnillo are capped by volcanic flows. Southwest of Fresnillo the Valdecañas range shows a succession of rhyolite flows and tuffs perhaps 500 m. thick, apparently younger than the tuff of Proaño hill, that may once have extended over the district.

The sedimentary breccia and the overlying volcanic rocks are little if any folded but locally are tilted as much as 20°. Large faults are not known at Fresnillo, but they cut similar volcanics elsewhere in the region.

No intrusive rocks are exposed at Fresnillo, but in the Plateros district 5 miles north of Fresnillo a small stock of granodiorite porphyry cuts rocks of the Proaño group.

The veins are younger than the Tertiary (?) tuffaceous sediments and probably younger than the rhyolite flows. At Plateros similar veins cut the granodiorite porphyry.

## ORE DEPOSITS

The ore deposits of Fresnillo are principally narrow veins yielding silver, lead, zinc, copper, and gold. Their primary ore minerals are pyrite, sphalerite (marmatite), galena, and chalcopyrite with minor pyrrhotite, arsenopyrite, pyrargyrite, and other silver sulphides. The chief gangue minerals, are quartz, calcite, and other carbonates.

The average width of the more important veins is a little more than one meter. They vary from well defined "formal" veins to irregular streaks.

At shallow levels, especially in the massive basal portion of the upper graywacke, the veins split up into many branches with irregular strikes and dips, and they are accompanied by disseminated silver-bearing sulphides in their wall rocks. The resulting stockwork type of ore deposit was mined for a width of about 200 m. in the glory holes of Proaño hill.

Replacement ore bodies are found in one section of the mine in the uppermost 25 to

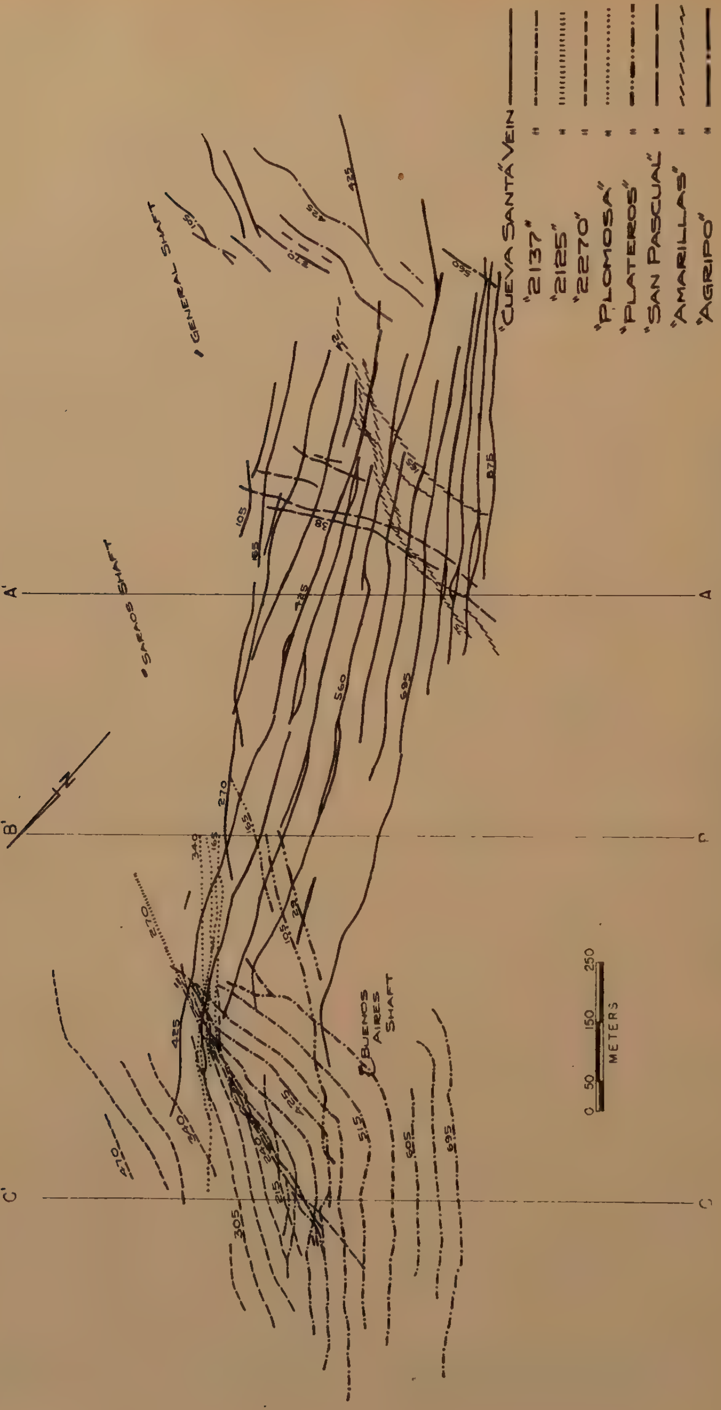


FIG. 1.—COMPOSITE PLAN OF PRINCIPAL VEINS, FRESNILLO.

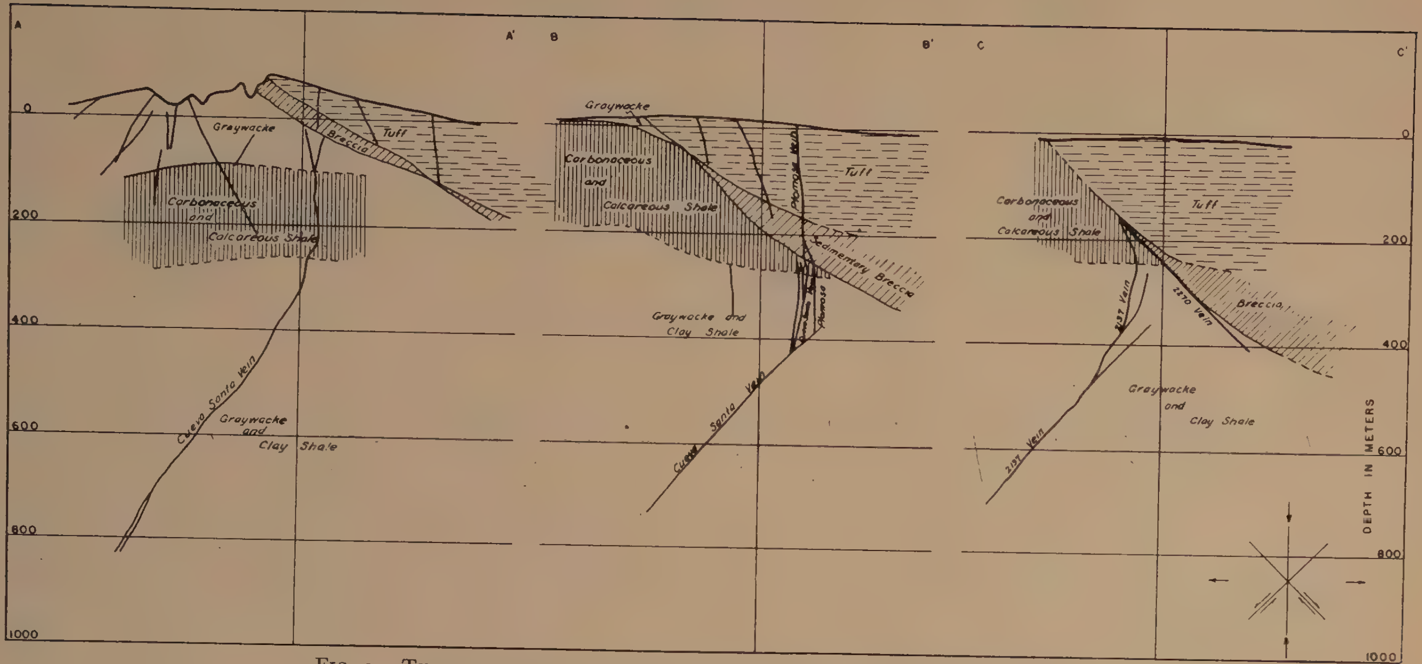


Fig. 2.—Three cross sections looking northwest on lines shown on Fig. 1.

40 m. of the carbonaceous and calcareous shale immediately below its contact with the overlying upper graywacke. These ore bodies are very irregular in outline both in plan and section and vary considerably in size.

Oxidation of the veins was general to a depth of about 100 m., the depth depending partly on the mineralogical character of the vein. The oxidized veins were attacked by the early miners, as their ores were amenable to amalgamation. The large-scale glory-hole operations started in 1919 mined remnants of the oxidized veins and the partially oxidized stockworks and disseminated ores surrounding them. Since 1927 the primary sulphide veins have been the most important producers. The mine workings now extend over a length of 2 km. Deepest work is on the Cueva Santa vein, which is being developed on the 920-m. level, and has been found by diamond drilling on the 965-m. level.

### *Vein Structure*

Some of the veins of Fresnillo, while narrow, are exceedingly persistent. The Cueva Santa vein, the most important vein known, has been followed for a maximum length of about 1000 m. on a single level and for a vertical distance of about 850 meters.

The veins usually are frozen tight to their walls with no slips or gouge seams or unhealed brecciation. They do not follow faults of any important displacement, but locally the vein fractures have been influenced by pre-existing surfaces of weakness, such as the unconformity at the base of the sedimentary breccia (see Fig. 2, section *CC′*).

Near surface the veins in the upper graywacke tend to be irregular in strike and dip. Veins are numerous in this environment and have many branches. Veins reaching surface in other formations (shale, breccia, or tuff) tend to have nearly vertical dips with occasional reversals. Their varying strikes and dips are characteristic of tensional fractures. At greater depths, principally in the lower graywacke, veins are fewer, more regular, and have flatter dips (40° to 60°). (Fig. 2.)

The form of the ore deposits and the pattern of the vein fractures depends partly on the character of the wall rock but seems to be even more closely dependent on depth.

The fracture pattern indicates deformation involving vertical shortening and horizontal extension. The diagram on the cross-sections of the veins (Fig. 2) indicates that the nearly vertical vein fractures can be considered as tensional openings. They occur chiefly at shallow depths. At greater depths (where loading was greater) or where guided by favorably oriented surfaces of weakness, the vein fractures have the attitudes of shear planes.

The veins were formed after considerable volcanic activity, probably in the Tertiary period. The regional geology shows no evidence of folding or of other large crustal movements of any kind near Fresnillo, but does show normal faulting elsewhere in the region.

Normal faulting under the influence of gravity is a common aftermath of regional volcanic activity. Such normal faulting is thought to have occurred at Fresnillo and in the surrounding region and to have formed the vein fractures.

### *Mineralogy and Paragenesis*

Since mining of oxide ore is now unimportant, and no complete data regarding the oxidized veins are available, this discussion of mineralogy will be limited to the sulphide ores.

The chief minerals of the sulphide zone are listed below in a suggested paragenetic sequence:

Quartz
Pyrite
Arsenopyrite
Pyrrhotite
Sphalerite

- Galena
- Unidentified silver mineral
- Chalcopyrite
- Silver minerals
    - Pyrargyrite
    - (Proustite)
    - (Polybasite)
    - (Matildite)
    - (Argentite)
- Calcite

Quartz, pyrite, and possibly calcite occur both early and late in the sequence.

Arsenopyrite and pyrrhotite follow after the early pyrite but are not present in large amounts and are not indicative of higher grade ore.

Sphalerite at Fresnillo is mostly of the iron-rich variety known as marmatite and varies from a dark brown to black in color. Analysis of a selected specimen of typical Fresnillo marmatite gave 10.5 per cent iron. Almost invariably, when examined under the microscope, the marmatite shows minute blebs and veinlets of chalcopyrite. The origin of this chalcopyrite is discussed briefly in a later paragraph.

Galena replaces sphalerite in a number of instances. Although the lead concentrate from the Cueva Santa vein contains a large proportion of the silver content of the ore, the mode of occurrence of the silver is not yet definitely established. In other veins, such as the 2125 and 2270, minute inclusions and veinlets of pyrargyrite and proustite in galena are of frequent occurrence. In specimens from the Cueva Santa, no such inclusions have been observed, nor have any definite occurrences of ruby silver minerals been noted along the parts of the Cueva Santa that have been worked in recent years. It has been suggested but not proved by other investigators of these ores that the silver content of the galena is due either to very minute inclusions of argentite or other silver sulphides, or to solid solution of silver in galena.

An unidentified mineral, possibly as yet undescribed, occurs in the lower levels on Cueva Santa vein. No detailed work has been carried out on this mineral, but a brief description will be given of it. Up to the present it has been recognized only on the 875 and 920-m. levels. In this section the vein is of light sulphide type with sparsely scattered ore minerals. Consequently, because of the difficulty of finding the unidentified mineral in contact with other sulphides, its age is somewhat uncertain; it appears to be later than the sphalerite and earlier than the chalcopyrite.

TABLE 1.—*Analyses of Unknown Mineral*[a]

| Constituent | I | II |
|---|---|---|
| Au | Tr | Tr |
| Ag | 3.00 | 3.76 |
| Pb | 48.00 | 54.04 |
| Zn | 0.40 | 1.30 |
| Cu | 0.00 | 0.24 |
| Fe | 4.60 | 2.27 |
| Mn | | 0.21 |
| Bi | 6.00 | 7.44 |
| As | 0.48 | 0.06 |
| Sb | None | 0.05 |
| Se | 2.00 | 1.90 |
| Te | | 0.11 |
| Sn | [b] | None |
| Ni | [b] | " |
| Co | [b] | " |
| Mo | [b] | " |
| V | [b] | " |
| S | 8.50 | 11.50 |
| CaO | 1.70($CaCO_3$) | 2.34 |
| MgO | | 0.14 |
| $Al_2O_3$ | None | 1.43 |
| Insol. | 19.60 | 11.20 |
| $CO_2$ | | 1.84 |
| | 94.28 | 99.83 |

[a] Analyses by C. W. Lawr, of The Fresnillo Company laboratory. The first is of a hand-picked concentrate of small fragments and the second a laboratory flotation concentrate of the mineral.

[b] A qualitative spectrographic analysis of the first sample indicated the presence of Sn, Ni, Co, Mo, and V, but no trace of these elements was found on analysis of the second sample.

Two analyses of this unknown mineral are given in Table 1.

On the assumption that the iron present is due to pyrite and the zinc to sphalerite, and deducting gangue impurities, the recalculated percentages of the more important constituents give the following composition for the mineral: Ag, 4.98 per cent; Pb, 71.70; Bi, 9.87; Se, 2.50; S, 10.95. Dividing these figures by the atomic weights of the elements gives the following formula: $Pb_{22}Ag_3Bi_3SeS_{22}$. Although this formula does not balance, and obviously is incomplete, it serves to indicate that the mineral, in spite of its homogeneous appearance in polished section, may be a mixture of galena with minerals such as guanajuatite $Bi_2(Se,S)_3$ and aguilarite $Ag_2(S,Se)$.

TABLE 2.—*Assays from* 2137 *Vein*

| Ore | Level, Meters | Au, Grams | Ag, Grams | Pb, Per Cent | Zn, Per Cent | Cu, Per Cent | Fe, Per Cent | Insoluble, Per Cent |
|---|---|---|---|---|---|---|---|---|
| Light sulphide........... | 270 | 0.90 | 1242 | 3.2 | 6.5 | 0.08 | 13.2 | 63.4 |
| Heavy sulphide.......... | 695 | 1.00 | 217 | 7.4 | 12.5 | 0.52 | 8.4 | 33.8 |

The unknown mineral is isometric, has galena white color and B hardness. Its reactions to the standard etch tests of Short are: $HNO_3$, area under drop turns brown and is slightly etched; HCl, area under drop stained brown; KCN, negative; $FeCl_3$, tarnishes brown to iridescent but only after long exposure (test of doubtful value); KOH, $HgCl_2$, and $H_2O_2$, negative. The polished section sometimes shows triangular pits like those in galena, but they are very scarce. In the hand specimen the mineral shows two good cleavages but lacks the cubical cleavage of galena and is slightly more bluish in color.

Possibly this unidentified mineral may be responsible for some of the silver content of Cueva Santa vein on the upper levels, since small amounts of selenium and bismuth have been detected in sulphide concentrates from the vein from as high as the 385-m. level. Because of its similarity in appearance to galena, the mineral might easily have escaped detection.

Chalcopyrite occurs in veinlets replacing all the minerals previously mentioned and is in turn replaced by pyrargyrite. The sphalerite contains numerous small blebs and inclusions of chalcopyrite, sometimes arranged parallel to the crystal planes of the sphalerite and sometimes apparently scattered at random. This type of occurrence has often been interpreted as evidence of contemporaneous deposition with the sphalerite and unmixing. In this case, however, since there is evidence of chalcopyrite later than galena and hence later than sphalerite, it seems more likely that the chalcopyrite inclusions are the result of replacement rather than of unmixing.

Pyrargyrite is the most common of the silver minerals. It replaces almost all the earlier formed minerals. Several other silver minerals are present in very minor amounts. They have been identified by Huber[8] as proustite, polybasite, and matildite. Argentite also is present.

The period of mineralization was brought to a close by the deposition of late pyrite, quartz, and calcite.

### *Mineralogical Types of Veins*

For milling purposes the sulphide ores are classified as light sulphide or heavy sulphide. Light sulphide is quartzose and/or pyritic ore valuable chiefly for silver and gold and low in sulphides of lead, zinc, and copper. Heavy sulphide is lower in quartz and also lower in pyrite, but high in galena and sphalerite and higher in chalcopyrite. Two representative assays from 2137 vein (Table 2), one of light sulphide ore at 270-m. level and one of heavy sulphide ore at 695-m. level, reflect the decrease in quartz and pyrite and in silver content, and the increase in lead and zinc at greater depth.

*Light Sulphide Ore.*—Light sulphide ore can be divided into five mineralogical types:

1. The Buenos Aires type occurs in the 2125, 2270, and upper parts of the 2137 vein. Its chief characteristics are fine, curly banding, fine-grained and sometimes chalcedonic quartz, plentiful pyrite, and a small amount of sphalerite and galena. Pyrargyrite, the chief silver mineral in this type of ore, is frequently visible.

2. The Refugio-San Pascual type is characterized by coarse quartz-carbonate vein filling with minor pyrite, sphalerite, galena, and pyrargyrite. Gold and silver are the valuable metals and are erratic and spotty.

3. The Catillas type was the chief type mined in the surface glory holes. It occurs chiefly in massive graywacke and consists of a quartz-carbonate gangue with pyrite, sparse sphalerite and galena, and unknown silver minerals. Disseminated silver-bearing sulphides occur in the graywacke walls.

4. The mineralized bed type is the ore of the replacement deposits in shale. It carries the same sulphides as the other types—pyrite, sphalerite, and galena—and is valuable chiefly for silver and gold. Carbonates and axinite make up much of the gangue.

5. The deep-level Cueva Santa type occurs at the west end of the lower levels of the Cueva Santa vein and has not yet been encountered anywhere else in the mine. Since it occurs below the heavy sulphide zone it is possible that other heavy sulphide veins may pass into this type at greater depth. The deep-level ore contains quartz, plentiful carbonates, pyrite, and sparse lead, zinc, and copper sulphides. The only silver mineral in the Cueva Santa vein that is sufficiently plentiful to be seen in a hand specimen is the unidentified mineral mentioned above. So far this mineral has been found only in the deep-level light sulphide zone.

*Heavy Sulphide Ore.*—Heavy sulphide ore is found in the Cueva Santa vein, the deeper part of the 2137 vein, and in other veins in the deeper part of the mine. It is characterized by relatively coarse galena, sphalerite, chalcopyrite, and pyrite in a subordinate quartz gangue. Although frequently it is fairly well banded, the coarse sulphides may mask the banded character in the hand specimen. The nature of the silver-bearing mineral in this type of ore is still uncertain. In a few places at deeper levels on the Cueva Santa vein diopside-hedenbergite and axinite occur in narrow veins paralleling the sulphide bands in the lode.

It is to be noted that all the mineralogical types of veins contain the same assemblage of ore minerals, with the exception possibly of the silver sulphides, but they differ from one another in proportions of minerals present and in textures.

### *Variations in Metal Contents*

Two veins at Fresnillo have been mined or developed over unusually great horizontal and vertical distances. These are the Cueva Santa vein and its branches, which are known for an over-all length of about 5000 ft. and a vertical range of 2800 ft., and the 2137 vein, known for about 2600 ft. horizontally and 2000 ft. vertically. Both these veins show notable changes in their content of silver, lead, zinc, copper, and gold over these developed areas. It is a matter of considerable interest in connection with the mining operation to understand the significance of this metal zoning, both to predict the behavior of these two veins at greater depths and to be able to anticipate similar changes in other veins that have not yet been developed over such a great vertical range. In this paper, however, the zoning is discussed only from the point of view of geological theory.

The Cueva Santa vein was worked in its eastern shallower portion by early operators. The ore in this section was oxidized in

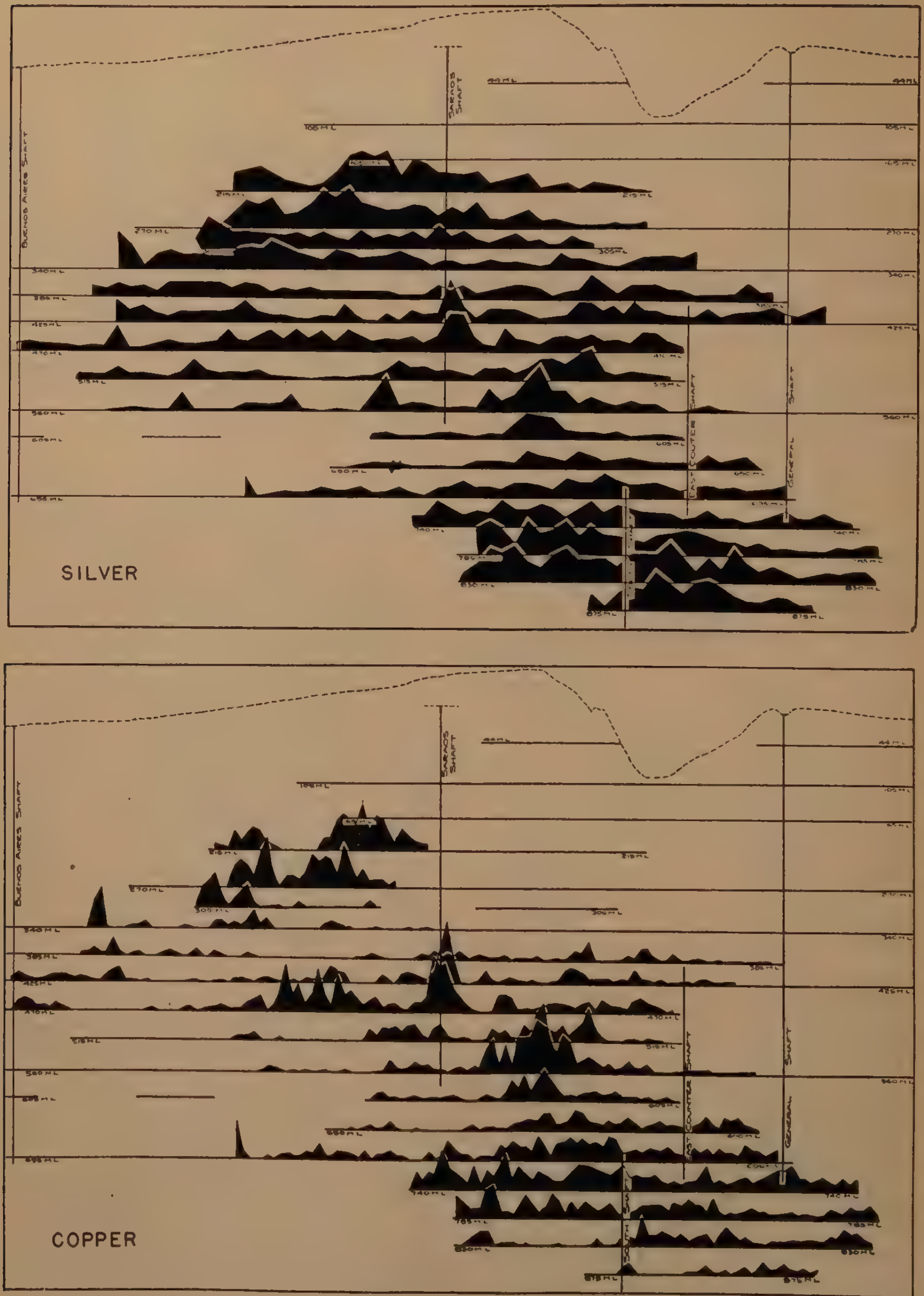
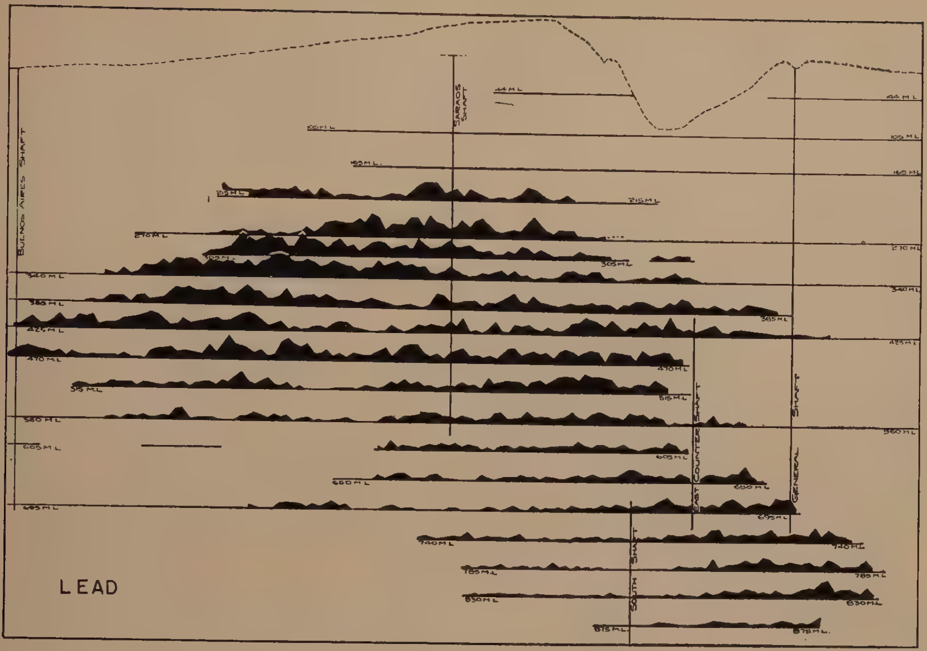
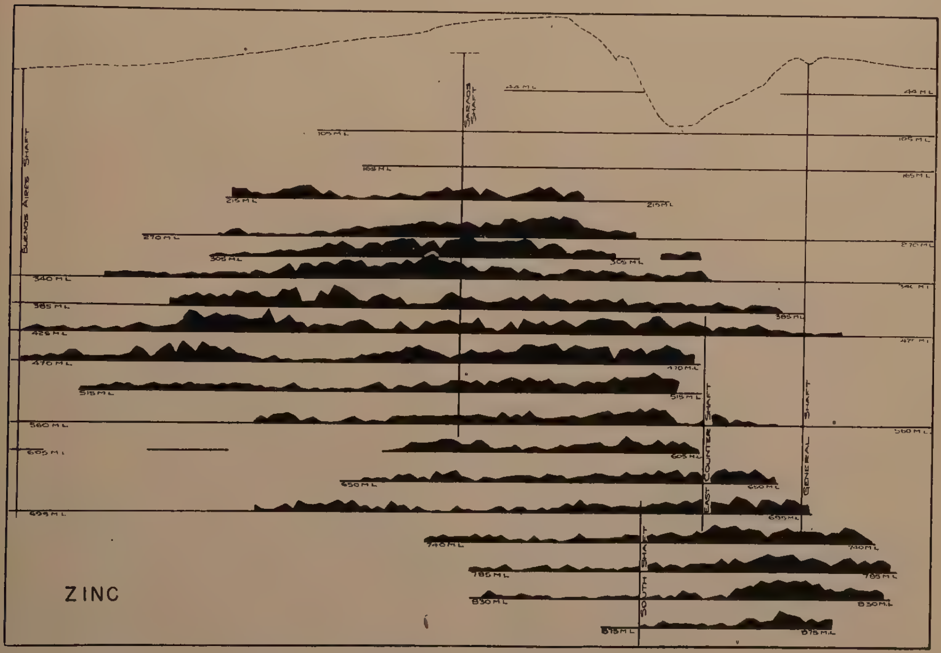


FIG. 3.—VARIATIONS OF METALS IN CUEVA SANTA VEIN





ALONG LEVELS ON LONGITUDINAL PROJECTIONS OF WORKINGS.

part but probably was of a high-silver, low-base-metal type originally, although it seems to have been a pyritic vein, rather than a quartzose one like 2137 in its upper portions. Cueva Santa vein where first discovered and worked by The Mexican Corporation in the western part of the mine was a heavy sulphide ore high in lead and zinc and relatively low in silver. From about 515 m. depth to the present bottom level at 920 m., however, the vein shows a marked change. A well defined zinc shoot and a narrower accompanying lead shoot pitch down toward the east in the eastern part of the vein (Fig. 3). In the western part of the vein a zone of light sulphide ore of a new mineralogical type (deep-level Cueva Santa type) appears. This ore is high in silver and low in base metals.

2137 vein shows a gradual transition from a light sulphide type of ore high in silver and low in lead, zinc, and copper into a heavy sulphide type lower in silver, high in lead and zinc, and irregularly higher in copper. Gold increases slightly, but its trend is not so definite.

The metal variations of the Cueva Santa and 2137 veins have been shown diagrammatically in two ways. Fig. 4 shows by graphs the average metal content of the two veins for each level, plotted against depth. The averages used in Fig. 4 represent the "ore shoot" or total minable length (not necessarily continuous) for each level. The assays used have been discounted to allow for sampling errors, and the averages have been weighted for vein widths.

Fig. 3 shows the variations along each level of silver, copper, lead, and zinc in the Cueva Santa vein on longitudinal projections of the workings on that vein. Since it was desired to show intensity of mineralization rather than economic value of the vein, the values used in Fig. 3 were averages of the undiscounted assays of the vein only for every 10-m. interval. In spite of the somewhat different bases for the two figures, both bring out the same features.

The following changes are noted in the graphs of Fig. 4:

In Cueva Santa vein silver is declining at the top of the graph, but with the lack of data above the 270-m. level it is not known whether this is a sustained decline or only a local irregularity. Throughout the heavy sulphide zone silver remains fairly constant, increases locally at 560-m. level, and reaches a minimum at 605-m. level. It then shows a rapid and fairly uniform increase to 875-m. level. Development below 875 m. is incomplete.

Lead and zinc are more or less uniform and equal as far as the 515-m. level, both decrease to the 560-m. level, then lead declines slowly, and zinc remains about constant but decreases slightly at the deepest levels. Zinc exceeds lead below 340 meters.

Copper parallels silver closely to the 740-m. level and then decreases while silver continues to increase.

Gold declines slowly but shows no pronounced trend. It is more nearly parallel to zinc than to any other metal.

In the 2137 vein the top of the ore shoot is below the 165-m. level. All development has been done by the Fresnillo Company, so that full data are available. The change from light sulphide to heavy sulphide ore takes place between 340 and 425-m. levels.

Silver is high at the top of the ore shoot and reaches a maximum at the 240-m. level. It then decreases rapidly to the 515-m. level and then declines very slowly to the 695-m. level. Development below 695 m. has only started.

Lead and zinc are parallel and increase rapidly from 215 to 340-m. level, with zinc greater than lead. Lead continues to increase to the 385-m. level and then remains fairly constant to the 695-m. level. Zinc is more erratic, being less than lead to the 470-m. level and then irregularly greater to the 695-m. level.

Copper is absent at the top of the ore shoot but increases gradually to a peak at the 425-m. level. From 425 to 695 m., it roughly parallels silver.

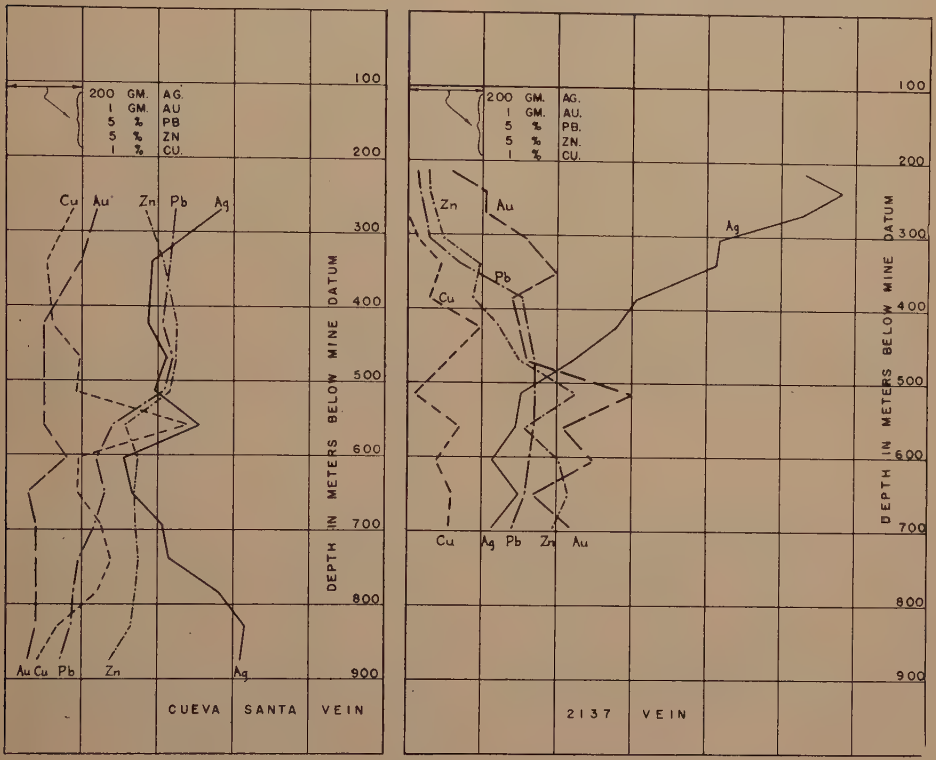


FIG. 4.—AVERAGE METAL CONTENT OF CUEVA SANTA AND 2137 VEINS.

Gold is more erratic and higher than in the Cueva Santa vein. It shows opposite variations to silver and copper and is more parallel to zinc.

Some other veins in the light sulphide zone show features resembling 2137 vein. The metal changes in the normal heavy sulphide zone of Cueva Santa vein from the 270 to 515-m. levels appear to be closely comparable to those in the heavy sulphide part of 2137 vein from the 425 to 695-m. levels.

The diagrams of Fig. 3 showing variations of metals in the Cueva Santa vein bring out the divergence at lower levels (below about 515 m.) into a zinc shoot and a narrower lead shoot at the east end of the vein and a high-silver shoot at the west end of the vein corresponding, respectively, to the normal heavy sulphide type of ore and the deep-level light sulphide type. Close comparison of the diagrams for different pairs of metals (this can best be made by superimposing a tracing of the diagram for one metal over that of another) shows that in the section above the 515-m. level the highs and lows for all four metals agree fairly well although there are some discrepancies and the actual ratio of the different metals may vary greatly. The same feature is brought out by the graphs of Fig. 4. Silver and lead agree fairly well down to the 515-m. level but show no correspondence below that level. Silver and copper show a rough parallelism to about the 740-m. level, although copper is erratic.

Lead and zinc correspond very well for all levels but at lower levels the lead shoot is shorter than the zinc shoot.

Longitudinal metal diagrams for 2137 vein, including one for gold not reproduced in this paper, show that silver varies with lead and zinc below the 515-m. level and with copper below the 560-m. level, but that at higher levels there is less correspondence, and in the light-sulphide zone none at all. Silver and gold vary together at all levels, and gold also corresponds fairly well with zinc. Lead and zinc fluctuate together at all levels. Copper is very erratic, showing only a slight tendency to agree with silver below the 560-m. level. In general the metals of 2137 vein fluctuate very roughly together in the heavy sulphide zone.

### *Conditions of Deposition*

*Age.*—The veins of Fresnillo cut tuffaceous sediments of probable Tertiary age and are believed to be younger than the rhyolites of the nearby Valdecañas hills. At Zacatecas and Sombrerete, veins, presumably of the same metallogenetic epoch, cut Tertiary volcanics that may be as young as Miocene.

The recency of the mineralization at Fresnillo is shown by the high thermal gradient still existing there, high rock temperatures and occasional small flows of hot water (about 135°F.) being found at a depth of 3000 feet.

*Depth.*—Since the veins were deposited, an unknown amount of rock has been eroded from above the present surface. Possibly the rhyolite flows and tuffs of the Valdecañas hills, now some 1500 ft. thick, once extended over Proaño hill, although the cover may have been less than that amount. Judging by the geological history of the district, it can only be said that the amount of overlying rock removed by erosion is more probably measured in hundreds of feet than in thousands.

The pattern of the vein fractures (Fig. 2) also supports the conclusion that the fractures, and hence the veins, were formed under light load.

We believe that the outcropping portions of the veins were most probably formed at a depth of less than 2000 ft. and possibly at less than 1000 ft. Several important veins did not outcrop but were first discovered at 300 to 500 ft. below the present surface.

*Mineralogical Evidence.*—The ore minerals and the vein textures of light sulphide veins of the Buenos Aires type clearly belong in Lindgren's epithermal group. Diagnostic features are their fine grain, fine curly banding, abundance of ruby silver, and the occurrence of occasional large vugs. The heavy sulphide ore might be classed as mesothermal or perhaps leptothermal.[9]

An anomalous feature in the mineralogy is the occurrence of abundant axinite and of diopside-hedenbergite, both minerals indicative of higher temperatures than usually are ascribed to epithermal or even mesothermal solutions. Axinite occurs at Fresnillo at least as high as the 100-m. level. In places it is found in altered shales remote from important veins, but its occurrence as an abundant gangue mineral in the replacement ore bodies in shale, and as parallel bands of axinite and hedenbergite in the Cueva Santa vein at lower levels, forbid any conclusion that the axinite belongs to an early period of alteration independent of the vein mineralization.

Heavy sulphide veins of the Cueva Santa group are cut in places by light sulphide veins, both of the Refugio-San Pascual and the Buenos Aires types. Nevertheless the 2137 vein, one of the Buenos Aires type, shows a complete and gradual transition within itself from light sulphide to heavy sulphide ore.

The relation of the deep-level Cueva Santa type of light sulphide ore to the heavy sulphide ore is different. This light sulphide may belong with a later generation of quartz that was deposited following reopening of the Cueva Santa vein, which in places encloses angular fragments of heavy sulphide.

*Conclusions.*—In most respects the Fresnillo veins can be classified as a telescoped sequence ranging from Lindgren's epithermal to mesothermal zone, or perhaps through Graton's leptothermal zone. Their actual known range is slightly more than 3000 ft., and they are believed to have formed beneath a cover of not over 1000 to 2000 feet.

The anomalous occurrence of high-temperature minerals, chiefly axinite, proves that the mineralizing solutions, at least in their earlier stages, were hotter than the usual epithermal or mesothermal solutions and indicate a relationship with Buddington's xenothermal[10] group of ore deposits formed at high temperature but at shallow to moderate depth.

In some districts mineral zoning is the result of successive waves of solutions of different composition that followed periods of opening or reopening of vein fractures. At Fresnillo the 2137 vein shows a complete transition upward from heavy sulphide to light sulphide ore. Specimens from this vein show the usual evidence of a paragenetic sequence of minerals, but there is no evidence that the change from heavy sulphide to light sulphide deposition was due to changes in character of the vein solution following brecciation or reopening of the vein fracture. The evidence in this vein is rather for selective deposition of different metals from a single solution under the control of changing temperature-pressure environments. The character of the wall rock did not affect the process.

The maximal deposition of ruby silver at the top of the ore shoots of the Buenos Aires type should be noticed. Similar silver bonanzas have sometimes been ascribed to secondary enrichment, but at Fresnillo there is no question of their primary nature.

It has been mentioned that heavy sulphide ore of the Cueva Santa vein and its branches is cut by light sulphide ore of 2137 vein. Each of these veins is believed to show a gradation within itself from heavy to light sulphide ore. When the younger vein, 2137, formed it deposited light sulphide at an elevation where Cueva Santa vein had previously deposited heavy sulphide, and the change from light sulphide to heavy sulphide in 2137 vein takes place 300 to 500 ft. deeper than in Cueva Santa. This may be explained as due to cooling at the source of the solutions and consequent deeper recession of the isotherms, but not by a change in composition of the solutions leaving that source.

The change at deep levels on Cueva Santa vein from heavy sulphide to light sulphide (deep-level type) appears, however, to be the result of reopening of the vein fracture, and can be explained in at least two ways. One explanation would be that this deep-level light sulphide mineralization is the youngest of all and was conditioned by a still further recession in depth of the isotherms. The alternative explanation is that the unidentified silver mineral containing bismuth and selenium that appears in this deep-level ore was capable of deposition at a higher temperature than the ruby silver of the shallower light sulphide types.

The control of the mineral and metal variations in the veins of Fresnillo is independent of kinds of wall rock and of kind of vein opening. We believe the control was the temperature-pressure environment, which changed with time at any one place. The pressure at any place within the vertical range of known ore deposition must have depended chiefly on nearness to the surface existing at the time. This surface was probably roughly parallel to the present surface and not far above it. The temperature at any place depended on the position of that place with respect to the main channelways leading from the source of the heat and of the mineralizing solutions to the surface. We assume that the source of the heat and of the solutions was roughly the same and that heat was carried into the sedimentary rocks of Proaño hill by the solutions.

## Acknowledgments

We are indebted to Mr. J. H. Ashley, General Manager of The Fresnillo Company, for permission to publish this paper and for assistance in its preparation. Several fellow geologists have given us the benefit of their comments.

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# The Alleged Mineral Zoning at Mount Isa

By Roland Blanchard,* Member A.I.M.E.

(New York Meeting, February 1943)

## Abstract

Recent discovery at Mount Isa of copper mineralization in commercial amount at a depth of 1000 ft., coinciding with downward termination of ore at about the same depth within the larger silver-lead-zinc lenses that have been the standby of Mount Isa's production to date, has been cited as an example in support of mineral zoning.

A previous paper by the author and Graham Hall[1] furnished a detailed description of the geology of the ore at Mount Isa as exposed up to that time, but necessarily dealt mainly with the silver-lead-zinc mineralization. Most of the geologic facts that constitute the first half of the present paper were set forth in the previous one in much greater detail, with full supporting evidence for the facts cited. The large amount of underground development (approximately 20 miles) and the extensive exploration carried out by diamond drilling (100,000 ft.) at Mount Isa, added to extensive microscope work conducted over a period of years, both by the company's staff and by others, had made possible the calculation within fairly close limits of mineralogical composition of the principal mineralized block of ground down to 1000-ft. depth, and had established closely the mineral succession, with reasons for deposition of the various minerals in the patterns now occupied by them.

The geologic facts as outlined in the previous paper remain as stated; but because substantial development within the new copper ore body and further exploratory diamond drilling within the new silver-lead-zinc area (Racecourse) have taken place since that paper was written, statements and figures relating to those two areas in the present paper supersede those furnished in the previous one. The recent work likewise has established more clearly the paragenetic relationship between the chalcopyrite and galena. Evidence confirming the relationship is furnished in the present paper.

The facts of ore occurrence at Mount Isa show

Accepted Zonal Succession
Silver sulphantimonides
Galena
Sphalerite
Tetrahedrite
Arsenical sulphides
Chalcopyrite
Pyrrhotite

Mount Isa Succession

| | |
|---|---|
| Chalcopyrite | In the main but probably not wholly contemporaneous |
| Silver sulphantimonides | |
| Tetrahedrite | |
| Galena | |
| Pyrrhotite | |
| Sphalerite | |
| Arsenical sulphides | |

They show further: (1) that instead of the silver-lead minerals yielding to increased zinc in depth, the zinc yields to increased silver-lead; (2) no dovetailing of the silver-lead-zinc sulphides into chalcopyrite takes place; (3) despite the large amount of pyrrhotite deposited in the district (estimated at upward of 100,000,000 tons), its deposition took place under moderately low temperature conditions; (4) no evidence has been found for increase in temperature of deposition, with depth, to deepest exploration in ore (1475 ft.). The anomalous condition at Mount Isa, both as to the unorthodox mineral succession and the largely unorthodox pattern of mineral distribution, is explained in the present paper,

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Manuscript received at the office of the Institute Nov. 13, 1942. Issued as TP 1652 in Mining Technology, November 1943.

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[1] References are at the end of the paper.

and is shown to be the natural and inevitable outcome of conditions that prevailed during the period of ore deposition.

Based upon the sulphide occurrences and sequences existing at Mount Isa, certain observations are made regarding the premises upon which the zonal theory rests. Although in some districts the sequence of sulphide deposition as postulated by the zonal theory seems unassailably established, it is suggested that the premises themselves may be less securely based than is commonly believed.

## Introduction

Continuous stoping of silver-lead-zinc ore at Mount Isa from the surface to a depth of approximately 1000 ft.; irregular downward termination of the ore near that depth within the larger bodies that have been stoped; and appearance at or near the same horizon of copper ore of stoping grade, which has been proved to an additional 350-ft. depth, with no sign as yet to suggest that it will not persist deeper; constitute a set of conditions that has been interpreted as representing mineral zoning. The following summarized description of ore occurrence at Mount Isa is presented to clarify the position.

## General Geological Setting

The Mount Isa deposit occurs within a 2-mile thickness of evenly bedded pre-Cambrian shale strata sandwiched conformably between quartzite formations, about ½ mile from its west edge. The beds strike north-south and in general dip 60°W.; but in vicinity of Mount Isa the shale strata, subjected to a thrust from the south-southwest, have been partly rotated and doubled back upon themselves, between their more rigid quartzite retaining walls, to produce folded structures of intricate and complex pattern. Associated with the overthrust folding is extensive faulting, and much minor crenulation and lenticular shearing. Shearing is confined mainly to bedding planes of the shale. The affected area has a length of 5 miles parallel to strike of the formations, attains a central width* of 2000 ft., and tapers irregularly therefrom toward either end. Disturbance is greatest within the broader central portion of the 5-mile length. Within that portion, occupying at any given elevation an area not exceeding 7000 by 1500 ft., and for convenience called the central mineralized block, occurs all ore of the district that has proved commercial to date.

The shale strata are noteworthy for their thinness; many of the beds are less than 1/500 in. and few of them more than ½ in. thick. Groups of dominantly thin beds in widths of 70 to 200 ft. alternate with groups of dominantly thicker beds of equal or greater width, but either type of bed is interspersed to some extent irregularly through the groups of the other. The thinner strata are mostly siliceous and slightly graphitic; the thicker ones are often, though neither necessarily nor dominantly, dolomitic.

Igneous rocks do not outcrop in the vicinity, and have not been encountered in underground exploration to a depth of 1350 ft., or in diamond drilling to a vertical depth of 1780 feet.

## Silver-lead-zinc Mineralization

Silver-lead-zinc mineralization occurs as replacements by sulphides of certain shale strata, mostly the thinner members. It began along sheared bedding planes and thence penetrated into the rock on either side, forming semi-massive to nearly massive sulphide layers, normally 1/100 to ¼ in. thick. Bands of ore sulphides alternate with, and to an important extent replace, other bands of pyrite-pyrrhotite; and sulphide bands of all types alternate indiscriminately with barren shale seams of equal or greater thickness; giving the ore a banded or ribboned appearance. Individual ore lenses range in size from bodies of a few thousand tons to one with proved stoping length of nearly 2000 ft., maximum stoping width of 250 ft., and maximum stoping depth exceeding 1000 ft. The lenses are mostly unsymmetrical in longitudinal outline, with metal content diminishing irregularly along their borders.

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* In this paper "width" is used to express horizontal distance; "thickness," to express stratigraphic distance.

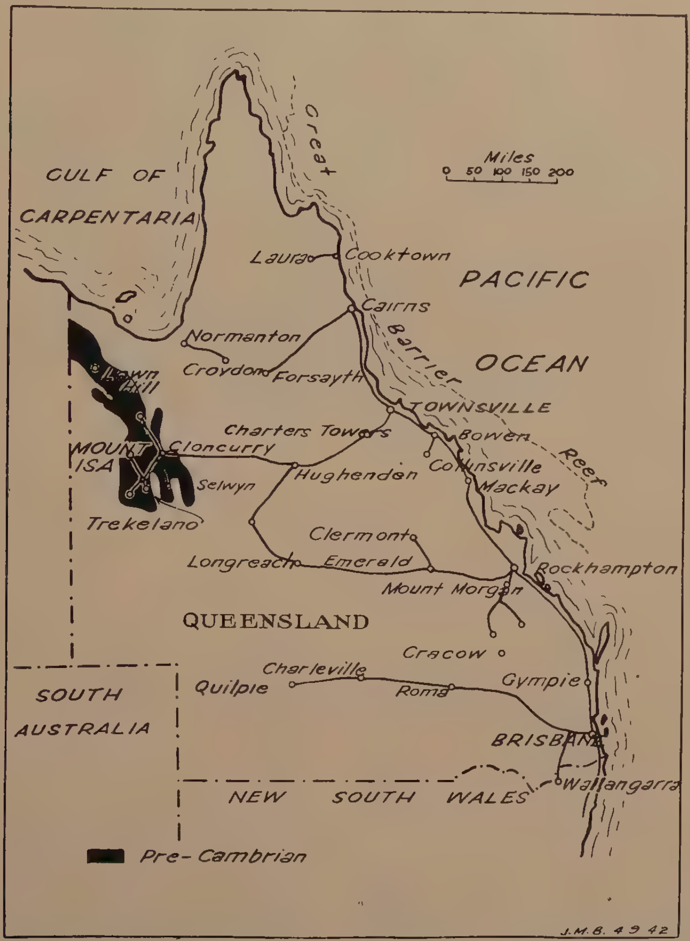

FIG. 1.—LOCALITY MAP, SHOWING ALSO THE PRE-CAMBRIAN OF NORTHWESTERN QUEENSLAND.

Processes that effected the mineralization comprise three well-defined events:

1. Major overthrust movement, accompanied by brecciation, followed by invasion of silica.

2. Crushing and fracturing movement, followed by invasion of coarsely crystalline "dolomite."

3. Main shearing movement, and deposition of first, second, and third generations of sulphides.

### *Major Overthrust*

The overthrust movement, with which is associated all major folding and faulting of the district, together with widespread crenulation among the thinner strata, was accompanied in its later stages by formation of the breccia zones. These comprise lenticular belts of strongly crushed and brecciated shale, which parallel in a general way the strike and dip of the shale strata, but cut across the beds at an acute angle, usually with steeper dip. The zones are confined mainly to the groups of thicker strata, and attain widths up to 600 feet.

Brecciation was in part accompanied by wholesale invasion by silica, but for the most part this followed the brecciation. The silica infiltrated its way into and through the brecciated material, more or less replaced the disturbed shale fragments, and converted the entire mass into a consolidated and rigid whole. The breccia zones have stubbornly resisted erosion, and remain today the chief topographic expression of Mount Isa's mineralization, rising to several hundred feet above the surrounding broad, flat valleys. Though impressive topographically, they are almost wholly barren of the silver-lead-zinc ore.

### *Crushing and Fracturing Movement*

Renewal of the thrust caused intricate crushing and thorough fracturing of the silicified breccia zones. This was followed, in turn, by large-scale invasion by coarsely crystalline "dolomite,"[*] which sealed up the fractures of the breccia, replaced much of the adjacent material, and reconsolidated the ruptured masses.

The "dolomite" reticulates irregularly through the shattered breccia, and occurs in all forms from scattered veinlets to nearly complete replacement bodies 50 ft. or more across. Locally it invaded certain strata outside the breccia zones, where it effected replacements up to 65 per cent of the rock mass over widths of shale up to 300 ft. across. Such thorough replacement has been disclosed as bodies of more than 1,000,000 tons.[†] When replacement by dolomitic minerals approximates two thirds of the total volume of such bodies and the bodies individually exceed 15,000 or 20,000 tons, they are called massive "dolomite" bodies.

Many such bodies are present. All exhibit a tapering upward, but none of the larger ones reaches the surface except as thin stringers. Impressive as they are individually or in the aggregate, they are greatly overshadowed by the breccia zones. It is estimated, for example, that to 1000-ft. depth within the central mineralized block total volume of the massive "dolomite" bodies is only one seventieth that of the breccia zones. Likewise, despite the usually higher concentration of dolomitic minerals within them, it is estimated that substantially more than 90 per cent of all coarsely crystalline "dolomite" of the district occurs in rocks outside the massive bodies, largely within the breccia zones. The point should be noted because it bears directly upon the argument for or against mineral zoning.

### *Main Shearing Movement*

*First Generation of Sulphide Deposition.*—Following deposition of the coarsely crystalline "dolomite," the thrust was again renewed. Its force however was receding, and the movement expressed itself chiefly through gentle but pervasive shearing along bedding planes of strata that had escaped consolidation by silicification or invasion of "dolomite." The beds most affected naturally were the groups of dominantly thin strata between or adjoining the breccia zones; and among them, in particular, the very thin (1⁄500 to 1⁄10 in. thick) more graphitic members.[*]

Gentle nature of the shearing may be judged from the fact that: (1) much of it is observable only under the microscope (see Figs. 6 and 7), and (2) seldom is a plane of shearing found to transect a crenulation. The shearing in this case might be likened to the leaves of a book gliding gently over one another as pressure s applied to one of its covers. The pervasive character of the shearing may be judged from the fact that the shearing persists along the 5-mile length of rock disturbance, and extends generally into all parts of that length, including locally unsilicified portions within the breccia zones themselves.

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[*] "Dolomite" is a term used to describe the intimate and characteristic Mount Isa carbonate gangue intermixture consisting of dolomite ankerite manganiferous siderite and minor calcite, with dolomite constituting nearly 80 per cent of the total.

[†] Long tons. To convert to short tons multiply by 1.12; to convert to metric tons multiply by 1.01605. This applies to all references to tons in this paper.

[*] Mount Isa shale averages approximately 0.40 per cent fixed carbon; the host rocks to the silver-lead-zinc ore bodies average 0.53 per cent fixed carbon.

Shearing of the first generation was succeeded by introduction of the district's fine-grained pyrite. Because replacement began along sheared bedding planes, and penetrated thence into the rock on either side instead of proceeding along a broad lateral invasion front, each bed retained its identity and the replacements assumed the characteristic banded pattern (Figs. 2 and 3). Replacement by the fine-grained pyrite took place throughout the 5-mile length, and pyrite of the first generation alone accounts for more than two thirds of all sulphides precipitated within the district. To depths explored, its content is estimated at more than 500,000,000 tons.

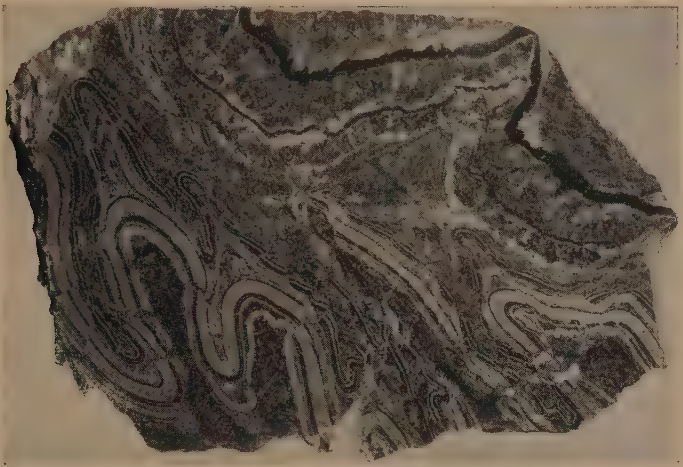

FIG. 2.—REPLACEMENT BY FINE-GRAINED PYRITE.

The light gray crenulations in bottom half of photograph represent the minutely thin-bedded shale strata replaced by fine-grained pyrite of the first generation. Although replacement is later than the crenulation, the pyrite faithfully replaces the strata selectively, irrespective of their position or shape. Irregular white specks in rude bands in upper half of photograph represent galena of the second generation. About two thirds natural size.

The pyrite penetrated also along numerous minute cracks or fractures that had developed irregularly in the breccia and massive "dolomite" bodies contemporaneously with the shearing, but its deposition within such masses was sporadic and on the whole lean, in no sense comparable to its widespread banded deposition along the sheared strata.

A characteristic of the first-generation pyrite, as previously suggested, is its fine grain size; the grains mostly range from 0.005 to 0.025 mm. in diameter. The mineral further is distinctive in that between 3 and 4 per cent of its content consists of pyrrhotite in solid solution.

Associated with later stages of the pyrite deposition occurred precipitation of similarly fine-grained sphalerite, representing possibly 5 per cent of the district's content of that mineral, but insignificant in tonnage as compared with the pyrite. With the sphalerite were precipitated also very minor amounts of chalcopyrite as ex-solution blebs. No silver or lead sulphides were deposited during the first generation.

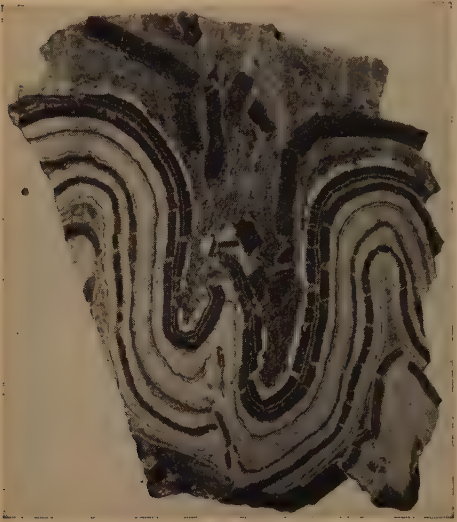

FIG. 3.—ANOTHER EXAMPLE OF REPLACEMENT BY FIRST-GENERATION FINE-GRAINED PYRITE OF THE MINUTELY THIN-BEDDED SHALE STRATA.

In this photograph individual beds 0.05 in. thick may be discerned, each retaining its individuality despite its nearly complete replacement by fine-grained pyrite. The upper half of the photograph shows how the more massive beds (0.1 to 0.5 in. thick) more commonly yielded to pronounced fracture and brecciation during the major overthrust movement.

About two thirds natural size.

*Second Generation of Sulphide Deposition.* Shearing along bedding planes of the shale strata, expressing the waning phase of the overthrust movement, persisted interruptedly and in gradually diminishing degree almost to the end of the three generations of sulphide deposition.

Following the close of the first generation, a period of quiescence had ensued. When shearing was resumed, the impetus behind it obviously had diminished; its development assumed a more lenticular pattern than its almost stratified regularity along the 5-mile length during the first generation, and the new lenses of shearing were confined more emphatically to the highly disturbed and previously most weakened section of ground, which constitutes or immediately adjoins the central mineralized block.

Major sulphides deposited were pyrite, sphalerite, pyrrhotite, galena, with successive overlaps and in the order named. Each replaced all preceding sulphides, while at the same time effecting replacement of the newly sheared mineralized strata adjoining.* Grain size of the sulphides, though still small, showed a general increase over that of the first generation; the diameter of the pyrite grains, for example, averaged about 10 times larger than those of the first generation. More than 90 per cent of the district's sphalerite and probably 75 per cent of its galena were deposited at this time. Although deposition of pyrrhotite, no longer admixed with the pyrite, was widespread and overlapped from the sphalerite cycle well into the galena cycle, deposition of sphalerite had ceased before that of galena began.

Because sulphide deposition is closely proportional to the degree of shearing in the silver-lead-zinc bodies, and because intensity of the shearing was waning throughout the second generation, the galena, taken as a whole, penetrated to a slightly lower height vertically, and to a slightly shorter distance laterally, along most of the areas of lenticular shearing than did the sphalerite. The result has been a generally higher zinc content near the upper and lateral peripheries of the ore bodies.

*Third Generation of Sulphide Deposition.* Almost to the close of the second generation, shearing along bedding planes had constituted the principal physical expression of rock disturbance associated with sulphide deposition. As an accessory effect of the overthrust, its locus necessarily had lain to the south-southwest. With westerly dipping beds, the direction of shearing therefore had been clockwise.

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* It is characteristic of the shearing movement throughout the three generations that important shearing usually did not take place along thicker beds until adjoining thinner ones had been sheared and replaced by sulphides.

Toward the close of the second generation, the locus of thrust began shifting, by gradual stages, to a deep-seated point more directly beneath or slightly into the footwall of the ore bodies; and although shearing did not wholly cease, dominant expression of rock disturbance thenceforth involved the formation of many small cross fractures. At its beginning, during late second generation, the fracturing expressed itself only as local minute "eggshell" crackled patterns. But as force of the thrust from the new locus increased, innumerable small but distinct cross fractures were produced, usually from a few inches to a few feet long and with only insignificant displacement, which cut indiscriminately across both barren shale strata and the existing mineralized bands. They were especially well developed in vicinity of existing silver-lead-zinc lenses. These more strongly defined and widely distributed cross fractures constituted the dominant and characteristic type of rock disturbance and deformation throughout the third generation. With westerly-dipping beds, and with the locus of thrust now lying more nearly beneath or slightly into the footwall of the 5-mile mineralized length, the cross-fracturing movement thus expressed itself in a counterclockwise direction.

The changed direction of rock disturbance proved to be important; for once more sulphide deposition took place, following the same mineralogical sequence as before. Only minor amounts of pyrite, sphalerite and pyrrhotite were precipitated, however; the principal sulphide precipitated was galena, estimated at 25 per cent of the district's total. Although some of it was localized along dying phases of the bedding-plane shearing, especially in the vicinity of reopened strike faults, a large proportion of third-generation galena was precipitated along the cross fractures; and because the cross fracturing manifested itself less vigorously near the horizon of the present-day surface than it did at greater depth, the outcome was a not infrequent primary enrichment by galena within lower portions of the silver-lead-zinc ore bodies, and especially within the more deep-seated of such bodies, than at their higher elevations.

TABLE 1.—*Metal Content at Black Star Mine*

| Lode No. | Level | Tons | Ag, Oz. | Pb, Per Cent | Zn, Per Cent |
|---|---|---|---|---|---|
| 1 | 4–5 | 578,024 | 4.6 | 8.6 | 9.6 |
| | 5–7 | 504,318 | 5.7 | 10.8 | 5.7 |
| | | 1,082,342 | 5.1 | 9.6 | 7.8 |
| 2 | 4–5 | 1,411 109 | 5.5 | 7.4 | 10.9 |
| | 5–7 | 4,046,666 | 5.6 | 7.5 | 8.7 |
| | | (3,525,000 | 5.7 | 7.7 | 9.0) |
| | 7–9 | 990,350 | 6.7 | 8.1 | 6.6 |
| | | 6,448,125 | 5.7 | 7.6 | 8.9 |
| 328 group | 5–7 | 111,800 | 6.0 | 9.3 | 9.6 |
| | 7–9 | 853.590 | 8.6 | 9.4 | 9.1 |
| | | 965,390 | 8.3 | 9.4 | 9.2 |

Thus, because of special conditions of shearing, fracturing, and sequence of sulphide deposition, the Mount Isa silver-lead-zinc ore bodies present the anomalous condition of commonly decreasing in zinc, and increasing in lead, content with depth. Since silver content of the ore is associated mainly with tetrahedrite, which was deposited contemporaneously with the galena during the second and third generations, silver content within the ore bodies likewise commonly increases with depth (within limits of existing exploration).*

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* The condition discussed deals solely with primary sulphide mineralization; it has no reference to secondary leaching and enrichment processes, which at a subsequent time operated independently within 200 ft. of the present surface.

The condition is well illustrated by Table 1, which lists silver-lead-zinc content for the different lodes, between the different extraction levels, at the Black Star mine. The Black Star mine has contributed more than 80 per cent of Mount Isa's silver-lead-zinc tonnage production to date.

In No. 2 lode, increase in silver-lead content for the 5 to 7 levels over the 4 to 5 levels is small if the full tonnage is included. In this case, however, it became possible to reach out into footwall of the lode to mine a block of more than 500,000 tons of low-grade ore, which normally could not have been stoped profitably but which in this instance became extractable because most of the level and extraction development costs had been already charged against the higher grade ore in its hanging wall (see Fig. 12). With the low-grade footwall ore excluded, and the comparison confined to parts of the lode stoped elsewhere, the figures for the 5 to 7 levels are 3,525,000 tons averaging 5.7 oz. Ag, 7.7 per cent Pb, 9.0 per cent Zn.

*Minor Sulphide Deposition.*—In addition to the major sulphides enumerated above, several minor ones other than the tetrahedrite were deposited.

Among them was arsenopyrite, which was precipitated during each of the three generations, always closely though not exclusively identified with the pyrite, and not thus far found overlapping into the galena cycle of either the second or third generations. Of the district's arsenopyrite, 75 to 80 per cent is estimated to have been precipitated with the fine-grained pyrite of the first generation.

Marcasite occurs as a minor sulphide during the second and third generations; chiefly, though not exclusively, as the hypogene alteration product of pyrrhotite. Its deposition during the first generation has not been established.

Polybasite, pyrargyrite and proustite, with abundance in the order named, were precipitated contemporaneously with galena of the third generation. They have not been identified as second-generation precipitates, but as yet it is not wholly safe to state that they were not deposited to some extent during that period. Sternbergite ($AgS.2FeS$), with its age not clearly defined, is another minor silver mineral. All of these complex sulphides combined, however, probably do not account for much more than 5 per cent of the district's silver. Tetrahedrite remains the dominant carrier of that mineral in the Mount Isa ore bodies.

As noted under discussion of the first-generation sulphides, chalcopyrite occurs as minute ex-solution blebs coprecipitated with sphalerite. It occurs similarly in the second generation. The amounts involved are small, however; most of the chalcopyrite deposited within the silver-lead-zinc bodies is a coarser product, with grain size up to 1.5 mm. and averaging probably 0.1 to 0.2 mm., which was precipitated contemporaneously with tetrahedrite during the galena cycles of the second and third generations. Its deposition may have extended over a slightly greater time range than did that of the tetrahedrite and galena; at least in the second generation, and possibly also in the third.

Estimated percentages of the various sulphides deposited during the three generations within the silver-lead-zinc bodies of the central mineralized block down to No. 7 level (approximately 1000 ft. average depth) are shown in Table 2.

*Associated Nonsulphide Minerals.*—Much of the fine-grained pyrite of the first generation has a translucent fine-grained quartz, intimately intergrown with it, as an interstitial mineral. No evidence has been found of the importation of quartz during the first generation. The interstitial quartz thus is regarded as representing in large part, and perhaps wholly, reprecipitation of indigenous $SiO_2$ released when the thin siliceous shale beds were replaced by the fine-grained pyrite. Less direct evidence of reprecipitation of such $SiO_2$ has been found for the second and third generations; but certain silica "flooding" effects, as distinguished from the interstitial type, in vicinity of large-scale sulphide precipitations during the second and third generations suggest that, locally at least, such solution and reprecipitation of $SiO_2$ may have been active during those generations also.*

TABLE 2.—*Estimated Approximate Percentage of Each Sulphide Deposited within the Silver-lead-zinc Bodies of the Central Mineralized Block Down to No. 7 Level (1000± Ft. Depth)*

| Sulphide | 1st Generation | 2nd Generation | 3rd Generation | Estimated Percentage of Total Sulphides |
|---|---|---|---|---|
| Pyrite | 85 | 10 | 4–5 | 29.67 |
| Arsenopyrite | 75–80 | 18–20 | 4–5 | 0.146 |
| Sphalerite | 5 | 92–93 | 3 | 37.45 |
| Cadmium sulphide |  |  |  | 0.103 |
| Pyrrhotite | 12–15[a] | 80–85 | 3 | 7.38 |
| Marcasite |  | 95–97 | 3 | 0.74 |
| Galena |  | 75 | 25 | 23.64 |
| Tetrahedrite (omitting Ag) |  | 65 | 35 | 0.165 |
| Polybasite, pyrargyrite, proustite |  | ? | 100? | 0.044 |
| Sternbergite, and other miscellaneous minor sulphides |  | ? | ? |  |
| Silver, segregated as metallic Ag |  | 60 | 40 | 0.047 |
| Chalcopyrite | 4 | 45–46 | 50 | 0.615 |
|  |  |  |  | 100.000 |

[a] In solid solution with the fine-grained pyrite.

Throughout sulphide deposition of the second and third generations there occurred once more extensive precipitation of "dolomite." This "dolomite" differs in physical characteristics, and in chemical and mineralogical composition, however, from the coarsely crystalline product.† Under most conditions the two types are readily distinguishable in the hand specimen. No firm evidence has been found for the introduction of extraneous dolomitic minerals during the three sulphide generations beyond a small amount of the coarsely crystalline product, known as "ladder dolomite," which is associated with similarly small amounts of vein quartz that penetrated chiefly along strike faults—all introduced at the beginning of the third generation. For reasons discussed in a previous paper,[1] the imperfectly crystallized, turbid "dolomite" associated with the silver-lead-zinc bodies is regarded as largely though not wholly a recrystallization product of the sulphide-replaced syngenetic dolomitic shale strata.* The fact that it occurs largely as peripheral fringes around the silver-lead-zinc or other sulphide bodies suggests, likewise, that it represents a precipitation product due to concentra-

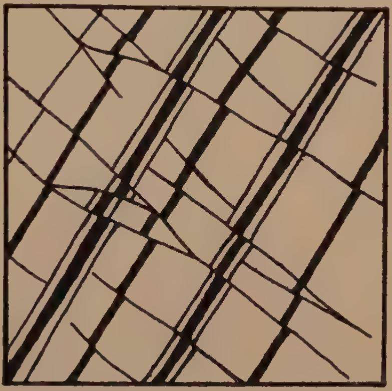

FIG. 4.—SECTION, LOOKING NORTH, SHOWING ANTICLOCKWISE THRUST UPON BEDS DURING THIRD-GENERATION CROSS-FRACTURING MOVEMENT. SECTION TO SCALE.

The vertical fractures are not shown. They point to the thrust having originated more directly from below (though still from the footwall position) than suggested by the section.

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* Source of the silica is less readily established in these cases because of difficulty in differentiating clearly between flooding effects brought about by silica indigenously recrystallized, and silica possibly imported with introduction of certain of the sulphides.

† Smaller and much less regular crystal form, usually more turbid color, and substantially higher percentage of true dolomite (see Table 3).

* Within the massive "dolomite" bodies a greater proportion of it has been recrystallized from the coarsely crystalline "dolomite," particularly where the latter has been extensively replaced by sulphides.

tion of $MgO$ in the solutions as result of the mineralizing solutions dropping a part of their sulphide lode.

Penninite was coprecipitated more or less consistently with sphalerite during the second and third generations, in an amount estimated at one fifteenth that of the sphalerite. It is found only in areas of shale penetrated by sphalerite, and is always closely associated with that mineral.

TABLE 3.—*Estimated Distribution and Mineralogical Composition of Mount Isa Dolomitic Gangue Minerals within the Central Mineralized Block Down to No. 7 Level*

| Type | Estimated Tons within Block | Estimated Distribution Based upon Analytical and Microscope Data | | | | |
|---|---|---|---|---|---|---|
| | | Percentage of Dolomitic Minerals within Block | Dolomite, Per Cent | Ankerite, Per Cent | Free Calcite Equivalent, Per Cent | Manganiferous Siderite, Per Cent |
| Syngenetic | 74,000,000 | 41.9 | 87.0 | | 8.5 | 4.5 |
| Coarsely crystalline | 68,500,000 | 38.8 | 62.373 | 30.5 | 0.127 | 7.0 |
| Recrystallized, turbid | 34,000,000 | 19.3 | 95.0 | 2.5 | | 2.5 |
| | 176,500,000 | 100.0 | | | | |

Graphite, as silvery spindles and radially fibrous shells 1 mm. to 1 cm. in size, occurs scatteringly in vicinity of strongly slickensided surfaces, usually at contacts of the massive "dolomite" with a shale bed that carries more than normal content of carbonaceous matter. Observed occurrences clearly were derived by segregation from the carbonaceous matter in the shale at time of the slickensiding, and nothing suggests association of the graphite with pegmatite-pneumatolitic phenomena.

Estimated proportions of all minerals deposited within the silver-lead-zinc bodies of the central mineralized block down to 1000± ft. depth are shown in Fig. 5.

From these statements of occurrence, and from the percentage and paragenetic relationships set forth in Table 2 and Fig. 5, the evidence thus suggests that, during the deposition of the silver-lead-zinc bodies at Mount Isa down to their developed depth of 1000± ft., no increase in temperature characterized the precipitation of sulphides at the lower horizons.

## COPPER MINERALIZATION

### *Distribution*

Up to the latter part of 1941, involving 6½ years of all-sulphide production at Mount Isa, annual average Cu content of the mill heads remained constant at 0.12 to 0.13 per cent. Tetrahedrite accounts for only a small proportion of the copper; most of it is present as chalcopyrite. Since silver-lead-zinc extraction has been proceeding gradually to greater depths during the period, a necessary deduction is that distribution of chalcopyrite through the silver-lead-zinc bodies down to No. 7 level has been essentially uniform.

Fig. 5 shows, however, that chalcopyrite is present within the central mineralized block in an amount far too great to be accounted for by the 0.12 to 0.13 per cent Cu of the silver-lead-zinc mill heads. For the block as a whole down to No. 7 level, content of chalcopyrite is estimated at nearly 62 per cent that of the galena.* Amount within the silver-lead-zinc bodies,

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* The estimate is not haphazard; it is based upon an exceptionally full sampling record, involving not only the usual underground headings, which penetrate at different elevations and in varying degree to nearly all sections of the central mineralized block, but is supported by results of approximately 100,000 ft. of exploratory diamond drilling within the block; of which the original 38 long surface drill holes, put down systematically along three successive deeper horizons, furnish more or less complete cross sections through the block along their respective coordinates of penetration.

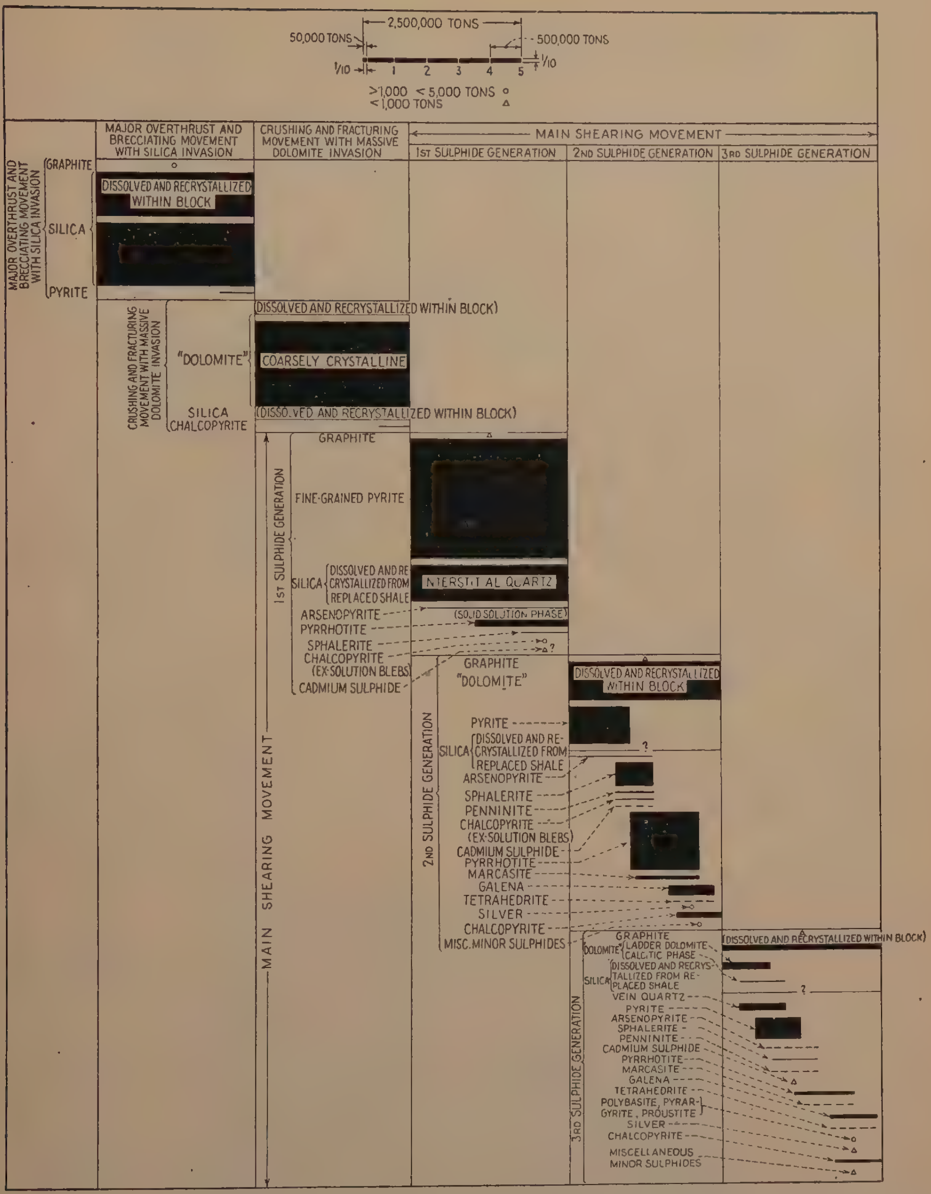

FIG. 5.—MINERAL DEPOSITION AT MOUNT ISA WITHIN THE 7000 BY 1500-FOOT CENTRAL MINERALIZED BLOCK TO NO. 7 LEVEL (1000 ± FEET AVERAGE DEPTH).

1. All minerals represent importations except where designated as dissolved and recrystallized products of indigenous origin.

2. Graphite shown represents recrystallized silvery type present within the ore bodies only; no adequate basis exists for estimating percentage of the recrystallized type elsewhere within the block. Nonrecrystallized graphite, being indigenous and unaltered, is not shown.

3. Proportional distribution of chalcopyrite at close of second and third sulphide generations represents an approximation only; further field evidence is needed before final estimate of the percentage distribution in these two instances is in order.

on the other hand, is estimated at only 2.6 per cent that of the galena, or 4.2 per cent of total chalcopyrite within the block.

It has been shown that silver-lead-zinc deposition was controlled mainly by degree of shearing along bedding planes of the thinner shale strata, although cross-fracturing of the third generation constituted an important locus for its precipitation during later stages of the shearing movement. In precipitation of the silver-lead-zinc minerals, the dominant control therefore was structural.

But in the precipitation of chalcopyrite, chemical control appears to have dominated. For the most part chalcopyrite is localized, not as replacements of shale strata, but of dolomitic minerals wherever found. It is estimated that even within the silver-lead-zinc bodies more than 50 per cent of the chalcopyrite has chosen a dolomitic host; outside those bodies, "dolomite" overwhelmingly constitutes the host.

Thus, just as extent of silver-lead-zinc mineralization in a broad way corresponds with degree of shearing along bedding planes of the shale strata, so chalcopyrite mineralization in a broad way corresponds with volume of the dolomitic minerals.

At Mount Isa, most of the chalcopyrite mineralization therefore is found, not within the thinner shale strata which localized the silver-lead-zinc bodies, but within the more massive strata where the latter have been strongly replaced by the dolomitic minerals, or importantly penetrated by reticulating seams of them. Most of the chalcopyrite at Mount Isa thus occurs within the massive "dolomite" bodies and breccia zones.

Since at Mount Isa none of the larger massive "dolomite" bodies outcrops at the surface, so that those bodies represent moderately deeply buried phenomena; and since they contain only an estimated 2.4 per cent of the total coarsely crystalline and recrystallized "dolomite" contained within the block down to No. 7 level, it follows, further, that most of the chalcopyrite deposition down to that level must have been precipitated within the breccia zones.

Within those zones its distribution occurs as minute veinlets, seams and disseminations, and as occasional small plums or pockets, for the most part far too erratically and sparsely distributed to be considered in any sense as ore; but nonetheless aggregating, in conjunction with its other lesser occurrences within the central mineralized block down to No. 7 level, an amount estimated at nearly two million tons.

Furthermore, the chalcopyrite's distribution is so generally uniform throughout most of the breccia zones in the first 1000 ft. of depth beneath the surface, that any 100-ft. vertical section thereof yields essentially the same amount of chalcopyrite as does any other 100-ft. section.*

As No. 7 level is approached, however, massive "dolomite" concentrations both within the breccia zones and outside them assume greater size and greater structural dominance in their respective positions within the block; and localized somewhere along irregular peripheries of those larger concentrations, but for the most part lying below their tops, occur the higher grade precipitations of chalcopyrite, which give promise of developing into commercial ore shoots.

The accident of distribution for massive "dolomite" in large bodies or in strong concentrations in vicinity of 1000-ft. depth at Mount Isa thus seems in the main responsible for the beginning of commercial copper mineralization near

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* At Mount Isa oxidation is more or less complete for 175 to 200 ft. beneath the surface. Amount and distribution of oxidized copper minerals, and of the limonitic pseudomorphs yielded by copper minerals wherever the latter have been leached within 200 ft. of the surface, testifies to chalcopyrite having been deposited originally through rocks of the present oxidized zone in much the same proportion and pattern as in the rocks directly beneath.

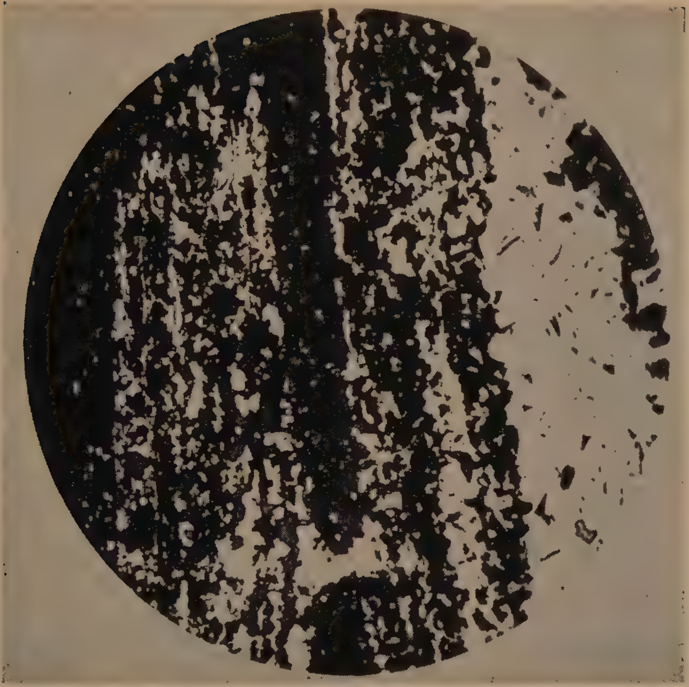

FIG. 6.—EXAMPLE OF SHEARING. × 42.

Specimen showing how shearing along bedding planes of slightly dolomitic shale (black) guided fine-grained galena replacement (white) during the second generation; and how a re-opened shear during the third generation (right of photograph) permitted deposition of third-generation galena in much broader bands than had occurred theretofore, thus effecting primary lead enrichment within such portions of the ore body, during the late third generation. (*Courtesy Australasian Institute of Mining and Metallurgy.*)

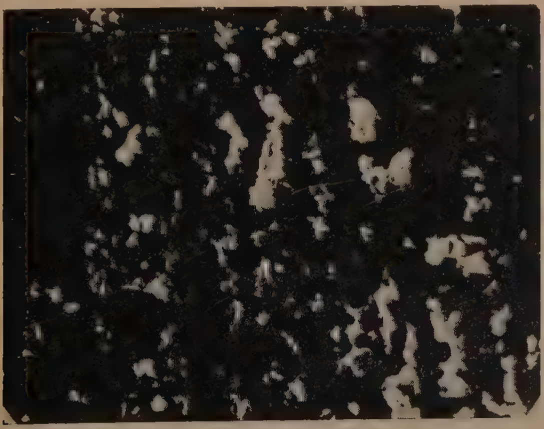

FIG. 7.—PART OF FIG. 6, SHOWING AT HIGHER MAGNIFICATION HOW BEDDING-PLANE SHEARING ALONG SHALE STRATA GUIDED DEPOSITION OF SECOND-GENERATION GALENA. × 92.

Galena grains here occur in widths of less than 0.005 mm. (*Courtesy of Australasian Institute of Mining and Metallurgy.*)

that horizon, rather than any inherent change in character of the mineralization at base of such silver-lead-zinc bodies as are becoming exhausted, through stoping, near that depth.

In view of the lack of evidence of change in temperature of deposition within the silver-lead-zinc bodies through the first 1000 ft. of depth below the present surface; and in view of the fact that nearly two million tons of chalcopyrite is distributed with an approach to uniformity through the central mineralized block within that 1000-ft. vertical depth; it thus would appear that had larger concentrations of massive "dolomite" occurred at higher elevations, commercial ore shoots of chalcopyrite probably would have become localized at those higher elevations also—independent of the silver-lead-zinc bodies but presumably extending over the same vertical range.

### *Age Relationship of Chalcopyrite to the Silver-lead-zinc Deposition*

Certain graphic and near-graphic intergrowths of chalcopyrite with the coarsely crystalline "dolomite" leave no reasonable doubt that chalcopyrite was deposited contemporaneously with the coarsely crystalline "dolomite" during the latter's invasion at the time of the crushing and fracturing movement, probably toward its close. The percentage of such chalcopyrite that can be identified with certainty is small, however; it is estimated at less (perhaps substantially less) than 5 per cent of the district's total.*

The relationship of the principal and later chalcopyrite to the silver-lead-zinc mineralization within the latter ore bodies during the first, second and third generations of sulphide deposition has been discussed. Outside the silver-lead-zinc bodies, however, age relationship of the principal and later chalcopyrite deposition has not been as specifically determined.

Some of the chalcopyrite outside the silver-lead-zinc bodies must have come at the time the ex-solution blebs were being deposited contemporaneously with sphalerite during the first and second generations. Most of it, however, obviously came late during the second generation. Evidence to support that assumption is that for the most part the chalcopyrite replaces both fine-grained pyrite of the first generation and the later and coarser pyrite and pyrrhotite, together with small amounts of sphalerite and galena, of the second generation. In turn it is cut by the sulphide-bearing cross fractures of the third generation, which at various places have been traced unbrokenly from the silver-lead-zinc bodies into the chalcopyrite-bearing massive "dolomite" and breccia.

It has been stated that the dominant control in precipitation of chalcopyrite was chemical, because the dolomitic minerals so overwhelmingly constitute its host. But wherever chalcopyrite is present in concentrations above the average, and particularly in amounts adequate for commercial exploitation, there occurs conspicuous development of many minute fractures, which facilitated ingress of the solutions or other medium that brought in the chalcopyrite. Such fractures mostly range from one millimeter to several inches in length, and ramify through the affected rock in a manner somewhat resembling glass shattered under strong pressure. From such minute shatterings the fractures show increase in size, contemporaneously with deposition of chalcopyrite, until in some instances both they and the chalcopyrite deposition almost merge with those of the third generation. The shatter zones are developed at a maximum near peripheries of the larger massive "dolomite" bodies, the shattering embracing both the strongly dolomitic portions and the siliceous breccia or other adjacent rock.[*]

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* As shown in Fig. 5, a small amount of pyrite was deposited, in part at least, even earlier. Both it and the early chalcopyrite may have been forerunners of the main mineralization which came later; but their close relationship to the main mineralization is not well established. In any event, amounts of both the early pyrite and the early chalcopyrite that may be identified positively as such are small.

From these occurrences and relationships the following deductions may be drawn:

1. The minute shatterings that localized concentration of chalcopyrite to form copper ore shoots in the massive "dolomite" bodies in the vicinity of and below No. 7 level are a part of the "eggshell" crackling movement, which at the same time was localizing the primary silver-lead enrichments at the close of the second generation in the adjoining thin-bedded shale strata. In both instances the crackling or shattering increased, until finally it grew into and merged with the cross-fracturing movement of the third generation.

2. During these shattering and cross-fracturing movements a small amount of chalcopyrite was precipitated generally through the silver-lead-zinc bodies, and a small amount of the silver-lead-zinc[†] minerals was precipitated more erratically, but nonetheless generally, through the chalcopyrite-bearing "dolomite," though each type adhered largely to its favorite host (see Table 7). As previously pointed out, field relationships suggest that deposition of chalcopyrite persisted slightly beyond that of the galena-silver,[‡] but not importantly, because even during the third generation galena and chalcopyrite are found consistently as coprecipitates.

3. Because in the chalcopyrite-bearing dolomitic areas no evidence appears of high-temperature minerals precipitated contemporaneously with the chalcopyrite (see page 22), it must be concluded that the chalcopyrite forming the copper ore bodies at and below No. 7 level (as far as exposed) was deposited with little or no increase in temperature over that which prevailed in the adjacent silver-lead-zinc bodies.

TABLE 4.—*Estimated Approximate Percentage of Total Chalcopyrite Deposited within the Central Mineralized Block Down to No. 7 Level*
(1000 ± *Ft. Depth*)

| | |
|---|---|
| Crushing and fracturing movement | <5 |
| Main shearing movement: | |
| 1st generation ex-solution blebs | 1/5± |
| 2nd generation ex-solution blebs | 4± |
| Close of generation | 60–75 |
| 3rd generation | 15–25 |

Difficulty in estimating closely the percentage of chalcopyrite precipitated during the late second and the third generations, respectively, arises from the fact that: (1) the amount of chalcopyrite as an ingredient of the third-generation cross-fracture fillings is highly variable from place to place, ranging from a fraction of 1 per cent to as much as 90 per cent; (2) thickness of the cross-fracture fillings within the copper areas likewise varies greatly, ranging from mere threads to 2-in. thick seams that may be almost exclusively filled with chalcopyrite; (3) extent of the cross fracturing itself varies markedly in different parts of the massive "dolomite" bodies and breccia zones, ranging from a single isolated occurrence to a modified stockwork pattern over faces 20 ft. or more across; (4) only a small proportion of the widely distributed copper-bearing ground has been penetrated by underground or stope headings, so that to date comprehensive exposures for visual observation (as opposed to diamond-drill cores) have been available at a few places only.

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[*] A crude stratigraphic alignment characterizes the shatter zones, suggesting that certain of the more massive shale strata were especially hospitable to replacement by the coarsely crystalline dolomitic minerals. But nowhere is the alignment rigidly defined; and, as Hall has pointed out, both massive "dolomite" bodies and the chalcopyrite concentrations associated with them often attain their maximum development along structural features, such as apexes of pre-"dolomite" folds in the shale. Their patterns likewise more often are craggy and projectile, like that of a molar, rather than conformable and banded, as with the silver-lead-zinc bodies.

[†] At many places the zinc is more sporadic in occurrence than is the galena and silver, pointing to its introduction at a time when the more uniform and sustained shattering toward the close of the second generation had not yet manifested itself strongly.

[‡] The principal evidence is that in the few instances in which the final surge of sulphide deposition can be identified in the presence of both galena and chalcopyrite, although both sulphides usually are found to be coprecipitated, the chalcopyrite occasionally floods the galena.

Many more exposures for observation may be needed before a trustworthy percentage estimate becomes possible. In the meantime, age distribution of chalcopyrite within the central mineralized block down to No. 7 level* is estimated, tentatively and somewhat fluidly, as shown in Table 4.

It is of interest to note that had not the locus of the rock deformational events at Mount Isa shifted toward close of the second, and more emphatically during the third, sulphide generations from clockwise shearing of the hanging wall to anticlockwise cross fracturing from the footwall; so as to rupture the previous banded sulphide deposition of the central mineralized block, and reopen the leanly mineralized lenses to admit, on the one hand, the enriching primary silver-lead sulphides to the silver-lead-zinc bodies; and to shatter certain portions of the massive "dolomite" bodies, and thereby admit, on the other hand, sufficient additional chalcopyrite into the generally leanly impregnated copper-bearing areas to raise their copper content well above that of the surrounding rock; grade of the ore in both cases probably would have been too low to permit extraction; and Mount Isa probably would not have developed into a major metal producer.

## Requirements of the Zonal Theory

The zonal theory postulates, as its ruling concept, that as deposition of sulphides or other ore minerals and their accompanying associates approaches the source of mineralization, the minerals separate out in a certain well-defined succession or sequence. With the changes of ore minerals, characteristic changes in the gangue and also in character of the wall-rock alteration manifest themselves.[2] The zonal arrangement may find expression in a lateral or vertical direction, or both. Deviations from the accepted sequence are conceded in deposits that were formed close to the earth's surface, or in depth at extremely high temperatures.

Those who accept the zonal theory are not agreed as to causes for the depositional sequence. Geologists, such as W. H. Emmons and Spurr, who relate sulphide* ore deposition directly to a magmatic source, conceive of the zoning as occurring upward and outward from one or more of the upper excrescences of a magma—either as illustrated in Fig. 8 or as some modification thereof which adheres to the essential relationships embodied in that figure. They conceive of the zoning as a phenomenon controlled mainly by temperature and pressure—minerals of the outlying zone being characteristically deposited at low temperatures, those of the intermediate zone at intermediate temperatures, those near the magmatic source at moderately high temperatures.

Geologists who refrain from committing themselves thus strongly to the magma-ore relationship accept temperature and pressure as probably important factors; but stress as equally important and perhaps as dominant such factors as relative solubilities, decreasing electrode potentials of the elements, decrease in sulphur content, and other physical constants of the respective sulphides. This permits them to detach the phenomenon from a direct magma relationship when and if desired. They visualize the relationship in general principle as set out in Fig. 9, whose only vital departure from that of Fig. 8 is absence of a visible magma.

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* Including the Black Star copper ore body down to No. 9 level, as far as exposed.

* For simplicity's sake, and unless otherwise specifically excepted, the term "sulphide" when used hereafter in discussion of the zonal theory will be held to include all hypogene minerals associated with the sulphides during deposition.

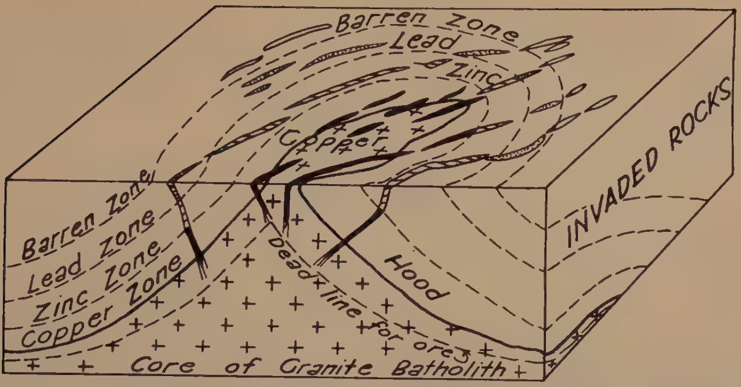
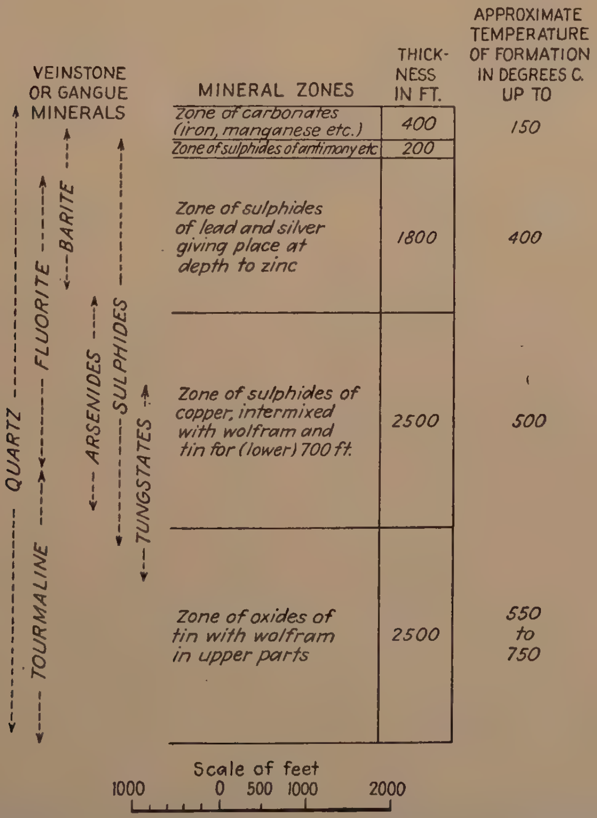


Fig. 8.—Diagram illustrating distribution of sulphides according to the zonal theory, on assumption of their direct derivation from a magma. (Emmons: Op. cit.,[2] p. 194.)



| MINERAL ZONES | THICKNESS IN FT. | APPROXIMATE TEMPERATURE OF FORMATION IN DEGREES C. UP TO |
|---|---|---|
| Zone of carbonates (iron, manganese etc.) | 400 | 150 |
| Zone of sulphides of antimony etc. | 200 | |
| Zone of sulphides of lead and silver giving place at depth to zinc | 1800 | 400 |
| Zone of sulphides of copper, intermixed with wolfram and tin for (lower) 700 ft. | 2500 | 500 |
| Zone of oxides of tin with wolfram in upper parts | 2500 | 550 to 750 |

Fig. 9.—Distribution of sulphides and other introduced hypogene minerals, according to the zonal theory as exemplified at Cornwall, England, but without showing relationship to the magma.

(After H. Dewey: *Proc.*, Geol. Assn., (1925) 36, 128.)

Irrespective of their adherence to or rejection of the concept of a direct magma source for the deposited sulphides, proponents of the theory are agreed that during deposition the sulphides separate out in the sequence set forth in Figs. 8 and 9.

Patterns of deposited sulphides in various mining districts are cited in support of the theory. Outstanding among them are the type examples of Cornwall (England), Silverbell (Arizona), and Zeehan (Tasmania). The pattern in a somewhat more modified or localized form is exhibited at Butte (Montana), Bingham (Utah), the Coeur d'Alene district (Idaho), Tintic (Utah), Bisbee (Arizona), Ely (Nevada), and several lesser districts.

As implied by previous statements, the zonal theory excludes freak relationships resulting from telescoping or other special effects brought about by successive periods or generations of sulphide deposition, or by distorted relationships resulting from a given sulphide passing out of its normal environment along a strong fault or similar structure. Absence of a specific sulphide or sulphides from the zonal column, as also barren or blank areas between successive sulphides, likewise is conceded for a given district. But the theory postulates that sulphides present, which belong to a single period or generation of deposition, should conform with the sequence. By implication, where continuity of sulphides prevails, more or less dovetailing of successive sulphides should be in evidence; likewise there should occur relative increase of a given sulphide in lower position with respect to its occurrence in the horizon above; otherwise mere parallelism or chance deposition of the alleged successive sulphides has taken place; which leaves the theory without support.

## THE FACTS AT MOUNT ISA

Because the Mount Isa ore bodies belong to the group classified as those deposited under moderately low-temperature conditions,* the only portion of the zonal column that need be considered is the one within that temperature range, as follows:[5]

Silver, 6—Argentite veins complex antimony-silver sulphides, stibnite, etc. Galena veins with silver. Commonly silver decreases with depth. Quartz gangue, siderite common, often increasing with depth.

Lead, 7—Galena veins, commonly with some silver. Sphalerite generally present, increasing with depth. Chalcopyrite common. Gangue is quartz and often carbonates (Fe, Mn, Ca).

Zinc, 8—Sphalerite veins with some lead and chalcopyrite, quartz gangue.

Copper, 9—Tetrahedrite veins, commonly argentiferous, chalcopyrite present. Some pass downward into chalcopyrite. Enargite (if present) generally with tetrahedrite and tennantite.

Copper, 10—Chalcopyrite veins, generally with pyrite, often with pyrrhotite. The gangue is quartz and in some places carbonates. Some pass downward into pyrite and pyrrhotite with a little chalcopyrite. Generally carry silver or gold.

Descriptions already supplied rule out, at Mount Isa, what proponents of the theory refer to as special or freak effects, except possibly for the two closely succeeding second and third generations of sulphide deposition. Since mineralogical sequence for those two generations is identical, save for probable but not fully proved absence of the complex sulphantimonides during the second, the theory may be held to be applicable for each of the two generations.

---

* The classification has been questioned by certain geologists unacquainted at first hand with Mount Isa conditions, because to explored depth along the 5-mile length of sulphide deposition pyrrhotite content, exclusive of that deposited in the solid solution phase during the first generation, has been estimated at more than 100,000,000 tons.[1] Mount Isa pyrrhotite, however, is distributed without special concentration and with an approach to uniformity throughout the 5-mile length, and is somewhat more extensive outside the ore bodies than within them, possibly because within the ore bodies it has been replaced to a greater extent by later sulphides. No increase in percentage of pyrrhotite with depth has been observed at the lower levels opened to date, and no characteristically high-temperature minerals have been found as depositional associates with the pyrrhotite at any place, except arsenopyrite in insignificant and widely distributed amount. The comprehensive microscope studies upon Mount Isa sulphides carried out by Grondijs and Schouten,[3] and by O'Malley and McGhie,[4] dispel doubt as to pyrrhotite at Mount Isa, either during the second or third generation, having been precipitated under other than moderately low-temperature conditions.

Disregarding for the present: (1) the nonsulphide gangue, which in most respects fits in satisfactorily with the accepted zonal succession, (2) the pyrite, which is not necessarily uncommon at any stage of deposition in most deposits, and (3) the small amount of chalcopyrite that was deposited during the early crushing and fracturing movement, and the small amount deposited as ex-solution blebs with the sphalerite, the condition at Mount Isa thus may be stated as follows:

ACCEPTED ZONAL SUCCESSION
Silver sulphantimonides
Galena
Sphalerite
Tetrahedrite
Arsenical sulphides
Chalcopyrite
Pyrrhotite

MOUNT ISA SUCCESSION
Chalcopyrite
Silver sulphantimonides } In the main but probably not wholly contemporaneous
Tetrahedrite
Galena
Pyrrhotite
Sphalerite
Arsenical sulphides

Using only the major sulphides, the successions are:

ACCEPTED ZONAL SUCCESSION
Silver-galena
Sphalerite
Chalcopyrite
Pyrrhotite

MOUNT ISA SUCCESSION
Chalcopyrite } In the main but probably not wholly contemporaneous
Galena-Silver
Pyrrhotite
Sphalerite

It has been shown that underlying reasons for the unorthodox depositional sequence at Mount Isa are: (1) components of the respective sulphides became available in the proper amounts and concentrations for precipitation at the appropriate times, (2) the shearing and/or fracturing phenomena provided suitable channels within the host rocks for entrance of the respective sulphides at the appropriate time and place, thus accounting for relatively more lead and silver near the bottom of most of the silver-lead-zinc bodies, and (3) in the case of chalcopyrite, which especially favored a dolomitic host, fracturing in the vicinity of the more massive and brittle dolomitic occurrences, toward the end of the second and during the third generations, provided the necessary channels to satisfy the requirements of that sulphide's depositional idiosyncrasies when chalcopyrite deposition was at a maximum.

No compelling reason is apparent why, under the conditions prevailing, sulphide deposition at Mount Isa should have followed another course. In the geological setting provided, the depositional sequence seems not only natural and logical, but inevitable.

As regards the dovetailing of successive sulphides at Mount Isa, it has been shown that in the silver-lead-zinc bodies down to No. 7 level average Cu content has remained constant during the 6½-year period of all-sulphide mill operation.*

Likewise Hall[6] has shown that, down to No. 7 level, any 100-ft. section in a vertical plane within the central mineralized block contains essentially the same total silver-lead-zinc content as does any other;

---

* To obtain trial mill runs in working out the necessary metallurgy, copper ore derived from development since September 1941 has been regularly put through the same ore pocket, ore bins, and concentration machinery as has the silver-lead-zinc ore. Dilution under such conditions is unavoidable, and Cu content shown in mill-head analyses for the silver-lead-zinc ore therefore no longer has significance in connection with the matters herein discussed.

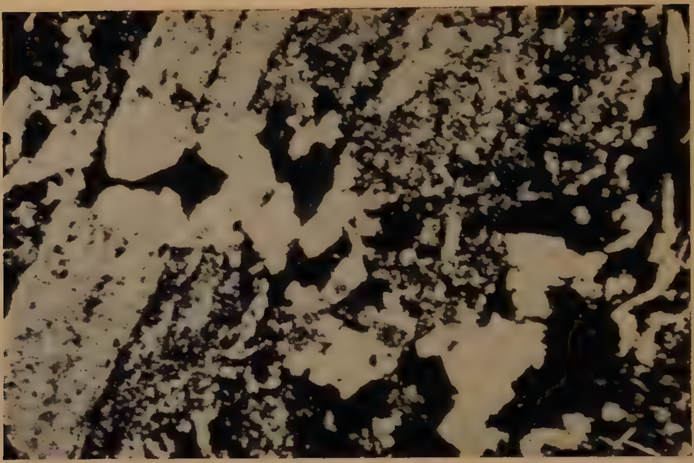


FIG. 10.—BANDED FIRST-GENERATION FINE-GRAINED PYRITE (LIGHT GRAY BAND CUTTING ACROSS LEFT HALF OF PHOTOGRAPH), WITH BANDED COARSE-GRAINED SECOND-GENERATION SPHALERITE-GALENA ADJOINING IT ON EITHER SIDE THROUGH REPLACEMENT OF THICKER-BEDDED SHALE STRATA. × 250.

Both are transected by still coarser-grained galena (with minor sphalerite) within a third-generation cross fracture that cuts centrally through the area. At right of photograph the cross fracture junctions with a third-generation reopened bedding-plane shear, which similarly is partly filled with galena and minor sphalerite. Gangue of the cross fracture and reopened shear is mixed vein quartz and "ladder dolomite" of the early third generation. Deposition of sphalerite was completed before deposition of galena began, both in the second and third generations. Sphalerite of the cross fracture and reopened shear occurs mainly along their edges, where its replacement by galena is not wholly complete. (*Courtesy Australasian Institute of Mining and Metallurgy.*)

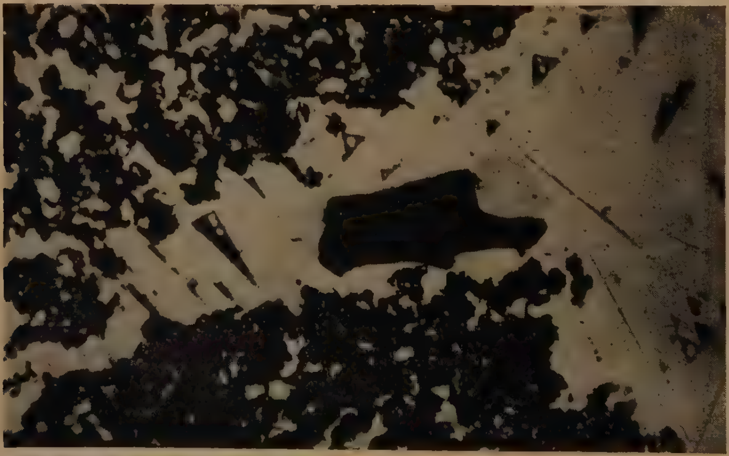


FIG. 11.—SECOND-GENERATION GALENA (SMALL IRREGULAR WHITE) REPLACING DOLOMITIC SHALE GANGUE, CUT IN TURN BY THIRD-GENERATION CROSS FRACTURE FILLED WITH GALENA THAT CARRIES POLYBASITE (PALE GRAY). × 55.

Polybasite, pyrargyrite and very minor proustite were coprecipitated with the galena. They thus came later than the sphalerite, later than most of the pyrrhotite, and on the whole contemporaneous with the chalcopyrite. (*Courtesy Economic Geology.*)

although, as pointed out in this paper, average zinc content tends to decrease somewhat, and average silver-lead content to increase somewhat, with depth, owing mainly to the proportionally greater amount of silver-lead minerals introduced in connection with the cross-fracturing movement of the third generation. Silver-lead-zinc distribution within the sheared thin-bedded shale strata, like the non-commercial scattered copper distribution within the breccia zones and massive "dolomite" bodies, thus has remained essentially constant in depth throughout the first 1000 ± ft. of the block.

As regards dovetailing of zonally successive sulphides at Mount Isa, the theory, therefore, finds little support in either the silver-lead-zinc or the copper deposits above No. 7 level.

But the question arises as to whether the appearance of commercial copper ore at and below No. 7 level may not constitute the needed proof that zoning begins at that level and becomes dominant at greater depth.

With ground of the central mineralized block below No. 7 level opened up by development of underground levels within the copper ore body only to a depth of 350 ft. (No. 9 level), and that only locally; and with exploration by diamond drilling in any type of ore to maximum depth of only 230 ft. below No. 9 level, likewise only locally; a conclusive answer cannot be given.

As of June 30, 1942, Black Star copper ore carried in the reserve as developed and prospective, between No. 7 and No. 9 levels, amounted to 1,430,000 tons of estimated 3.07 per cent Cu grade. That represents 127,000 tons of chalcopyrite.* Nowhere within so restricted a block of ground has a similar amount of chalcopyrite been disclosed above No. 7 level.

It has been noticeable, however, that diamond-drill holes that penetrate the breccia and massive "dolomite" bodies at and below No. 7 level no longer disclose the generally uniform distribution of chalcopyrite in small amount throughout those rocks that was common at higher elevations; concentration of chalcopyrite to yield lenses of 2 per cent to occasionally 5 per cent Cu ore are surrounded, in a striking number of instances, by breccia and massive "dolomite" with a noticeable decrease in chalcopyrite content compared with drill holes in similar rock at higher elevation. Thus it is not inconceivable that, in producing locally the concentrations of chalcopyrite that constitute ore at and below No. 7 level, nature may have done so by impoverishing the surrounding country at some distance from the copper ore bodies, without altering total chalcopyrite content within the central mineralized block for any given 100-ft. vertical section.*

Much additional and more intensive penetration of the chalcopyrite-bearing ground, both within the copper ore bodies and especially distant from them, is needed before an authoritative opinion upon this point may be ventured. It is even possible that no conclusive opinion may be in order until all of the ore-bearing ground between 1000-ft. and 2000-ft. depth at Mount Isa has been opened up as intensively as has been that between the surface and 1000-ft. depth.

---

* Diamond drilling in the Black Star footwall breccia zone at No. 7 level has disclosed, near a moderately small massive "dolomite" body contained within the breccia, a chalcopyrite-bearing lens, which may or may not develop into a commercial ore shoot. Drill holes laterally along the strike, and above and below it on the dip, have eliminated the possibility of a major ore shoot (Fig. 12). Chalcopyrite contained within the lens therefore can raise only incidentally the total amount of material classed as copper ore.

* In that connection it is worth noting that chalcopyrite carried in the ore reserve between No. 7 and No. 9 levels at the Black Star mine represents less than one fourth the amount estimated to be present throughout any 350-ft. vertical section in the non-ore areas above No. 7 level. It emphasizes how misleading, with respect to total content of a given sulphide within a district, may be the acceptance of only the portion classified as ore.

Whatever may be the eventual outcome in respect to the copper content, thus far there is no support, in character of the gangue minerals, to suggest increase in depositional temperature in vicinity of the copper ore bodies. The only noticeable alteration or variation in gangue minerals in the vicinity of such bodies is the partial recrystallization of the coarsely crystalline "dolomite," which was effected during the sulphide deposition. Such recrystallization has resulted in a 67 per cent increase in siderite at the expense of ankerite and dolomite, as shown in Table 5—surely not an argument either for increase in temperature of deposition or for prevalence of the moderately high temperatures with which hypogene chalcopyrite deposition usually is credited.

Contrary to the impression prevailing in some quarters, the silver-lead-zinc ore bodies at Mount Isa have not terminated in the vicinity of No. 7 level. At the Black Star mine, which contains the known commercial copper ore body, there has been termination merely of one lens (No. 1 lode), and approaching termination of another (No. 2 lode), which is the largest and most persistent lens of such ore that has been actively stoped at the Black Star mine to date.

At Mount Isa the silver-lead-zinc ore bodies mostly terminate downward when and because the zones of shearing, which localized such deposits, feather out along the hanging wall against either massive "dolomite" or breccia. But in exploration conducted to date it has been found that as one such ore body frays out in depth along the hanging wall, another appears in the sheared strata farther into the footwall, to take its place.

TABLE 5.—*Comparative Estimated Mineralogical Distribution of Dolomitic Gangue Minerals at Mount Isa*

| Body | Percentage Distribution | | | Ratio | | |
|---|---|---|---|---|---|---|
| | Dolomite | Ankerite | Manganiferous Siderite | Dolomite | Ankerite | Manganiferous Siderite |
| Massive "dolomite" bodies......... | 62.5 | 30.5 | 7.0 | 2.05 | 1 | 0.23 |
| Black Star copper ore body[a]........ | 59.8 | 28.5 | 11.7 | 2.10 | 1 | 0.41 |

[a] Based upon results from 22 diamond-drill holes through the Black Star copper ore body between No. 7 and No. 9 levels.

Under the zonal theory an increase in As content might be reasonably expected with depth. The copper ore body, however, yields less than one-half as much As per ton as do the silver-lead-zinc ore bodies; and, ironically enough, such change as is registered within the copper ore body is a decrease of As content with depth, as shown in Table 6.

But again the question arises whether, even though neither volumetric nor temperature increase may have taken place in connection with the chalcopyrite deposition in depth, mineral zoning is being expressed through termination of the silver-lead-zinc ore bodies as the copper ore-bearing section of ground, with depth, is being approached.

It thus happens that in the footwall of the largest actively stoped silver-lead-zinc lens (No. 2 lode), which is terminating not far below No. 7 level, there has been exposed another promising lens (328 lode), whose crest lies 700 ft. beneath the surface, and which shows its greatest width and highest grade ore where intersected by the deepest drill hole put down at Mount Isa to date. Its average silver-lead-zinc content is greater than that of the No. 2 lode, and it exhibits the characteristic increase in silver-lead over zinc content with depth that would be expected from the geological descriptions already furnished; but with no greater Cu content than prevailed in the other silver-lead-zinc ore at higher elevations.

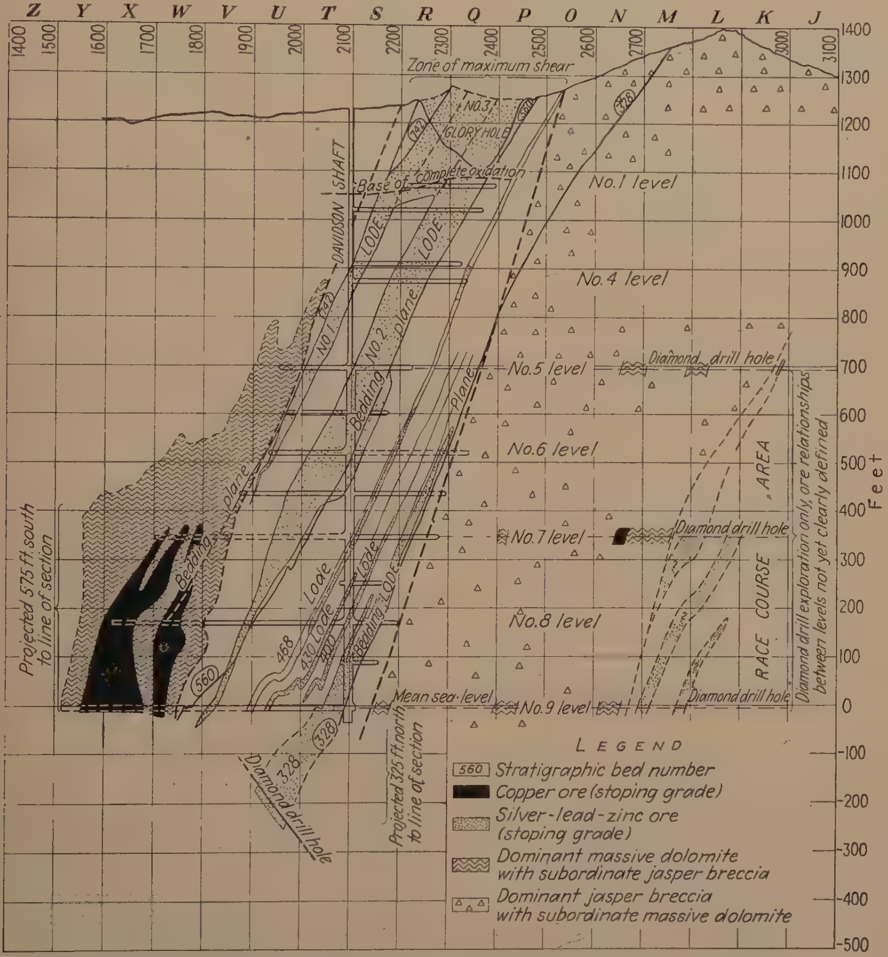


FIG. 12.—LOOKING NORTH THROUGH DAVIDSON SHAFT, BLACK STAR MINE.
Showing, with projection, relationship of the silver-lead-zinc and the copper ore bodies to massive "dolomite" bodies and breccia zone.

The conditions described are shown in Fig. 12, which is a cross section through central portion of the largest silver-lead-zinc lenses (Nos. 1 and 2 lodes) extensively stoped to date, but with the hanging-wall copper-ore lens projected 575 ft. south, and the footwall silver-lead-zinc lens (328 lode) projected 325 ft. north, so as to bring the three en echelon ore shoots into a single cross-sectional plane.

The more deep-lying silver-lead-zinc ore body (328 lode) shown in Fig. 12 is not a freak occurrence. Diamond drilling elsewhere in geologically favorable portions of the central mineralized block has disclosed other silver-lead-zinc lenses, between No. 7 and No. 9 levels, of the same general mineralogical composition and grade of ore as the one discussed above, such as those in the Racecourse area (Fig. 12). In no instance has there been noted a significant change in character of the gangue with depth.

Not all parts of the block between No. 7 and No. 9 levels that are regarded as favorable for the occurrence of silver-lead-zinc ore have been tested, and it is too soon to state whether the condition noted by Hall—namely, that for any given 100-ft. vertical section within the central mineralized block down to No. 7 level the amount of silver-lead-zinc mineralization is essentially constant—will persist to or beyond the greatest exploratory depth at which ore has been thus far intersected (1475 ft.); but existing evidence points strongly that way.

Similarly, the Black Star copper ore body, to No. 9 level, discloses no decrease in content of the small amount of galena present within that body, though a decrease in sphalerite is indicated, as might be not unreasonably expected from its known decrease with depth in the silver-lead-zinc bodies. Whatever change in galena is revealed by exploration within the Black Star copper ore body to date is in the nature of an increase in depth, as shown by the figures in Table 6, based upon diamond-drilling results.

Table 6.—*Accessory Metal Content of Black Star Copper Ore Body*

| Level No. | Number of Diamond-drill Holes for Which Complete Analyses Are Available | Average Percentages | | |
|---|---|---|---|---|
| | | Pb | Zn | As |
| 7 | 6 | 0.15 | 0.15 | 0.015 |
| 8 | 12 | 0.14 | 0.16 | 0.014 |
| 9 | 4 | 0.25 | 0.05 | 0.013 |
| | | 0.17 | 0.125 | 0.014 |

Further development of the copper ore body is needed before firm conclusions in this matter may be drawn, but thus far at least there is indicated, in the copper ore body as in the silver-lead-zinc ore bodies, no decrease in galena content with depth.

The Mount Isa deposit thus presents an instance in which not only is the succession of sulphides badly out of line with accepted standards of the zonal theory, but one in which the silver-lead-zinc and the copper depositions occur parallel to but on the whole independent of each other in a vertical plane, with each 100-ft. vertical section down to 1000-ft. depth carrying approximately the same amount of the respective sulphides as any other 100-ft. section. With respect to the silver-lead-zinc bodies the evidence points to a continuance of that condition to deepest existing exploration of 1475 ft. within those bodies. With respect to the copper bodies, the evidence is not yet clear, but a strong suggestion exists that the same conditions may obtain within those bodies also.

In the silver-lead-zinc mineralization the ore sulphides have been concentrated sufficiently, in a succession of en echelon lenses, to yield ore more or less uniformly from level to level from the surface downward. In the copper mineralization the ore sulphide has remained too diffusely scattered through the rocks to constitute ore until a depth of approximately 1000 ft. has been attained. Favored at that depth by more concentrated occurrence of a host rock for which the chalcopyrite has shown especial affinity, the previously widely dispersed chalcopyrite has become localized into lenses of sufficient size and grade for commercial extraction. Deeper development is needed to determine how uniformly, from level to level, and to what additional depth below the present proved 350 ft., such concentrations of chalcopyrite will persist.

The only tangible support for the zonal theory thus far found at Mount Isa is an increase to 0.04 per cent, in recent years, of As content of the silver-lead-zinc mill heads from the previous annual average of 0.03 per cent. It is largely neutralized by the indicated decrease in As content with depth in the Black Star copper ore body noted on page 22; and for a mineral present in so small amount and so widely dispersed as is the arsenopyrite at Mount Isa, and with 75 to 80 per cent of the mineral estimated to have been precipitated with the fine-grained pyrite during the first generation (see Table 2) before any silver-lead deposition, and before more than very minor amounts of zinc and copper deposition, had taken place, it seems a slender thread of evidence upon which to hang the zonal theory's application to Mount Isa. But the evidence is presented in fairness to proponents of the theory.

TABLE 7.—*Estimated Percentage Distribution of Sulphides in Various Bodies at Mount Isa*

| Sulphide | Total Sulphides, Per Cent | | | |
|---|---|---|---|---|
| | In Central Mineralized Block to No. 7 Level | In Ag-Pb-Zn Bodies to No. 7 Level | In Massive "Dolomite" Bodies to No. 7 Level | In Black Star Copper Ore Body between Nos. 7 and 9 Levels[a] |
| Pyrite | 75.73 | 29.67 | 33.331 | 30.239 |
| Pyrrhotite | 16.42 | 7.38 | 28.796 | 17.268 |
| Sphalerite, including isomorphous Fe. | 3.37 | 37.45 | 4.600 | 1.1105 |
| Galena | 2.13 | 23.64 | 1.946 | 1.1105 |
| Chalcopyrite | 1.312 | 0.615 | 29.548 | 49.195 |
| Marcasite | 0.850 | 0.740 | 1.4378 | 0.855 |
| Arsenopyrite | 0.1557 | 0.146 | 0.2657 | 0.166 |
| Tetrahedrite, (omitting Ag) | 0.0148 | 0.165 | 0.0456 | 0.056 |
| Cadmium Sulphide | 0.0092 | 0.103 | 0.0127 | |
| Silver, segregated as metallic Ag | 0.0043 | 0.047 | 0.0135 | |
| Miscellaneous minor sulphides | 0.0040 | 0.044 | 0.0037 | |
| | 100.0000 | 100.000 | 100.0000 | 100.0000 |

| | Central Mineralized Block | Ag-Pb-Zn Bodies | Massive "Dolomite" Bodies | Black Star Copper Ore Body |
|---|---|---|---|---|
| Estimated per cent, as sulphide content, of total tonnage within body or block | 19.64 | 41.97 | 4.52 | 18.01 |
| Estimated per cent of total tonnage of central mineralized block down to No. 7 level, comprised by | 100.00 | 4.198 | 0.52 | [b] |

[a] Based upon results from 22 diamond-drill holes between Nos. 7 and 9 levels.
[b] Below No. 7 level. Insufficient data at present for estimating comparative tonnage.

## CONCLUSIONS

In this paper no attack has been made upon the zonal theory. The author has first-hand acquaintance with a majority of the mining districts cited to substantiate that theory, and in three of them has spent from several months to several years mapping and studying the structural and mineralogical relationships in detail. He has observed the accepted standard of mineral succession too firmly established in a number of those districts to question the essential validity of the succession at such places.

That does not mean that he accepts such succession as necessarily a process in itself. Nor does it mean that he feels restraint in questioning whether the number of proved conformances to the succession is as great as the various citations in geologic literature might indicate.

It has been shown that, to the greatest explored depths at Mount Isa (variably down to 1475 ft. in ore), little relating to the character or distribution of the various sulphides supports, and most of it contradicts, the theory of mineral zoning. What the conditions may be below present explored depths cannot be forecast. Conceivably partial support for the theory may be disclosed at greater depth, though at least several irreconcilable features seem at the moment unlikely to be disposed of.

The chief point of interest, however, is that when viewed superficially—that is, without specific knowledge of the mineralogical relationships and detailed structural controls, and taking into account only the ore bodies instead of the total sulphide distribution scattered through the rocks within the principal ore-bearing portion of the district—an outwardly convincing case for mineral zoning at Mount Isa seems firmly established. When viewed with fuller knowledge of the facts, the case disintegrates.

Only a small number of the mining districts classed as type examples for mineral zoning have had the benefit of the day-to-day systematic recording of geological detail, from their outcrops down to the deepest underground workings or drill holes, and covering in a unified way the entire mineralized district involved, which since the inception of operations more than eleven years ago has been the settled practice at Mount Isa. Nor has such observation and recording always been supplemented by the exceptionally full and comprehensive series of sampling results, covering all phases of the mineralization phenomena, which have been available at Mount Isa.*

On the other hand, some of the districts cited to substantiate the zonal theory—by virtue of inaccessibility of much of the ground, and the resulting nature of attack upon the problem that could have been made since the zonal theory came into existence—necessarily have had to be judged mainly by the positional occurrence and compositions of their commercial ore shoots, with a wholly inadequate knowledge as to detailed mineralogical distribution or structural controls prevailing throughout the surrounding affected rocks.

In view of the ease with which such appraisal might have misled even the most fair-minded observer of short acquaintance at Mount Isa, it is but natural to consider the possibility of a similar error having been committed elsewhere in some of the accepted districts of mineral zoning.

At Mount Isa distribution of the sulphides in their unorthodox succession, and to an important extent unorthodox zonal pattern with reference to the sulphides' source, has been shown to be the result of: (1) components of the respective sulphides having been available in the proper amounts and concentrations for precipitation, at the particular time when (2) access to their eventual seats of deposition was provided by the immediately preceding or contemporaneous shearing or fracturing, supplemented, (3), in the case of chalcopyrite, by accessibility to a host of especial favorability at the precise period when volume of the ore fluid that conveyed the chalcopyrite was at its maximum. Had the crackling movement toward the end of the second generation, which later by natural progression merged into the cross-fracturing movement of the third generation, not occurred at that time, and had there been substituted, instead, further weakened shearing from the hanging-wall locus of thrust, it seems likely that most of the chalcopyrite that eventually went into formation of the copper ore bodies would have dissipated itself through the breccia zones, in much the same manner as it obviously did at higher elevations where such extensive bodies of massive "dolomite" were not present; and the concentrations of chalcopyrite, which in vicinity of 1000-ft. depth beneath the present surface have attracted so much attention at Mount Isa during the past year, never would have come into existence, to be seized upon by proponents of the zonal theory as another proof of their concept.

It thus would seem, from these considerations, that when the facts that control ingress of specific sulphides at specific periods to specific places in any given district are adequately understood, not only are the unorthodox successions and patterns of such district explicable, but they seem both the natural and inevitable outcome, irrespective of what may have been the disposition in some other district.

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* Among the so-called zonal districts, Butte is perhaps the outstanding example where such approach to the problem has been carried out systematically and comprehensively.

That brings into clearer perspective the question whether the controlling factors enumerated in the second paragraph above are not fundamentally the factors that should be stressed for determining, in any given case, the succession and pattern of mineral distribution for the deposit or mineral district, rather than stressing some hypothetical rule of succession. For it is far from established that all the sulphide and other mineral components that go into the formation of a complete ore deposit, even where only one generation of sulphide deposition prevails, are present at the source at any given time, or that they necessarily become available in a given order. Convenient as may be the idea, for many purposes, of the assumption of an omniparient and omnipresent ore fluid at any particular stage of sulphide deposition, the idea remains an assumption, which for the most part not only is lacking in direct evidence but which in many respects is inconsistent with results observed in numerous mining districts.

It seems realistic to assume that the reason so-called mineral zoning is expressed in various districts is primarily because the special conditions of ore fluid and precipitation-favorability operate more frequently in that sequence than do the ones in which some other combination prevails. But that is quite a different matter from saying that relative temperatures, solubilities, etc., as applied to the parental fluid for the whole sulphide sequence, are the controlling factors. The end product or pattern in many instances may be the same; but the controlling principle is entirely different, because it does not take a set of conditions that clearly are end results and interpret them as processes.

Manifestly, before final opinion may be expressed, much more needs to be learned about the multiform factors that control intermixing within the parental fluid, and the precipitation of sulphides under conditions not thus far subject to direct observation. But in view of the new light that conditions at Mount Isa have focused upon the problem of mineral zoning, and the several supposedly established factors relating to the zonal theory which thereby have been called into question; added to the lack of cohesion of viewpoint prevailing among proponents of the theory itself; one wonders whether the premises upon which the theory rests may not be less securely based than is generally acknowledged. At least it does not seem out of place to suggest that, in their present state, the prevailing ideas regarding the phenomenon of mineral succession in ore deposition might be more appropriately referred to, perhaps, as the zonal hypothesis.

## ACKNOWLEDGMENTS

Acknowledgments are due the Australasian Institute of Mining and Metallurgy and *Economic Geology* for permission to reproduce certain photographs used in illustrating this paper.

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# The Contact Pyrometasomatic Aureoles

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(New York Meeting, February 1948)

## Abstract

The definitions of the terms commonly used in connection with metamorphism at elevated temperature near igneous contacts are reexamined. As the study of tactite zones associated with igneous contacts progresses, more and more emphasis is put on the localization of the ore by structural forms. The generally early age of the silicates and late age of the sulphides is well established. Postsilicate structural deformation is likely to localize ore bodies. The silicates are often replaced by the ore minerals.

We may not be far from proving that the elements involved in the formation of silicate aureoles are merely normal "volcanic" emanations, that silica, lime, magnesia, and alumina, and probably soda and potash, are likely to undergo local transfers only. The work of Newhouse suggests the agents of the formation of sulphides in deposits of various classifications are the volcanic emanations.

## Introduction

The purpose of this paper is to define the terms commonly used in connection with the subject, to briefly review and discuss the broad features of the aureoles, to indicate some of the features lately recognized, to discuss the genesis, and to suggest how the associated ore deposits can best be studied and developed.

When developing a subject in mining geology one is soon conscious of the vast amount of pertinent literature and that most relevant ideas, no matter how obscure, have been considered. One can only hope to make some small contribution in point of view, a few confirmatory observations, or, perhaps, suggest a small new idea indebted to the most recent data. It is nearly impossible, also, to give specific credit, and adequate credit, to all the contributors to a given subject. This study, furthermore, was not intended to cover the field exhaustively, nor is the bibliography complete.

## Definition of Terms

There is much confusion in the use of the broad terms concerned. Needless to say, metamorphism[a] means change of form; metasomatism, change of substance. The term contact metamorphism, however, has been widely misused possibly by a majority of economic geologists and engineers. As originally adopted by European investigators it meant changes effected largely by heat (thermal effects) in the rocks invaded by plutonic intrusives, but did not include those local changes usually at the immediate contact brought about by the addition of substance, i.e., by metasomatism. The term has become clouded. In Europe it has

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The background for the subject as here treated is largely restricted to the Western Cordillera where many fine examples of contact pyrometasomatic aureoles have been described. The paper was prepared for and read at the meeting which founded the New Mexico Geological Society, April 12, 1947. Manuscript received at the office of the Institute Dec. 15, 1947. Issued as TP 2357 in Mining Technology, May 1948.

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[a] Some schools, notably Wisconsin, use the term in a very broad sense and even include weathering.

been extended to include regional metamorphism "miles from any igneous contact"[1]; in North America, to local altered zones around intrusives predominantly effected by metasomatism. Kemp[2] seems not to have made the matter clear. Lindgren[3] considered the various implications and proposed the general term pyrometasomatism[a] for changes by addition of substance within the temperature range of silicates, and contact pyrometasomatism for such changes near an igneous rock contact. Lindgren's terms are accurate but seem awkward to many. He also proposed the term pyrometamorphism for metamorphism at elevated temperatures. Many students who have studied the problem in the field have recognized both pyrometamorphism and pyrometasomatism in the contact zones, though it is the latter which is the most intimately related to ore deposits

A review of all the less general terms is needless here. The term *tactite* which, according to Grout,[4] refers to "contact-mineralized rock with various minerals formed from limestones or other soluble rocks by igneous emanations," is convenient and has been used extensively lately. Originally proposed by F. L. Hess it probably should be expanded to include all pyrometasomatic silicate rock regardless of the character of the host, or whether it is related to an igneous contact or not. The term *skarn* has a more limited meaning. It refers to garnet-epidote-pyroxene rock, probably the most common variety of tactite. *Garnetite*, of course, refers to a tactite in which garnet predominates. *Hornfels*, according to Grout,[4] is used for "dense, somewhat cherty-looking shale, silicified by contact action," but it seems difficult to justify a rock term that has a spatial implication. *Porcellanite*[4] probably should be extended to include all baked, i.e., "porcelainized," shales and limy shales. The broader use of hornfels as a textural term as outlined by Grout for dense, recrystallized sugary-grained rocks probably should stand.

## GENERAL FEATURES

The broad features of the metasomatic contact-silicate zones are, of course, well known. Their outstanding characteristic is that the host rock is usually relatively pure carbonate rock, but in quite a few cases igneous rocks are pyrometasomatized, in fact, are even garnetized. The shales and shaly carbonate rocks seem to be more sensitive to the purely thermal effects than to the metasomatic. Sandstones and quartzites are relatively inert to both processes. The selection of certain minerals for certain rocks may be very marked, but tends to be obscured where the mineralization intensity is high. In this case the final gross mineral composition is usually simple and often approaches the composition of an easily fusible slag. The volume of metasomatic silication in certain carbonate beds, usually the pure ones, may be on the order of ten times as great as in other beds.[a]

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[1] References are at the end of the paper.

[a] Lindgren says (Ref. 3, p. 696) when discussing pyrometasomatic deposits:

"Their mode of occurrence is metasomatic—that is, they are formed by replacement of the enclosing rock *with addition of substance*. (The italics are the author's.) They are formed at high temperature by emanations issuing from the intrusive. They are, therefore, suitably termed pyrometasomatic deposits."

[a] F. F. Grout in a personal communication strongly criticizes this statement as follows:

"'The volume of metasomatic silication' refers to a particular kind of change, but I see no sharp division between your examples. The *additions* from a magma to a shale are so near the composition of a shale that they are not easy to estimate. The same additions to a carbonate rock are easy to estimate. I am profoundly convinced there are large additions in contact change of shale to schist—think of the fluorine needed to make biotite and boron to make tourmaline. And think of the volume of contact schists, miles wide, as compared with measurements in feet, for the contact zones in limestones. Your bald statement 'on the order of ten times as great' needs a lot of evidence to support it."

My statement, perhaps, should be restricted to the non-schistose contact zones which closely resemble each other and which are common in the west. The beds are usually predominantly limestone with low dips. Doubtless schists receive moderate additions of substance, but one may ask if these changes are not often due to regional metamorphism. (See Harker[1]

The effect of structural forms, particularly faults and masses of breccia, is usually much more apparent after the type of detailed studies usually needed in ore finding have been made than has been expressed in the earlier literature. This phase of the subject has been discussed at some length by Knopf.[5] In general, one may say that the dependence of the mineralization—using this term in its general sense—upon channels such as joints, fractures, faults, and the like, for the transference of heat and substance, appears to increase as one progresses in time from the early metamorphic stage through the metasomatic to the sulphide ore epoch. The metasomatic silicates have usually been guided by faults and zones of breccia, often adjacent to the intrusive. The adjacent intrusive or so-called source intrusive, is usually proved older than the adjacent silication. I have never seen a case where the "source" intrusive cuts the ore and silicates. The silicates and ore normally cut the intrusive. One wonders if the metasomatic silicates localized on the contact *are* primarily genetically related to the adjacent intrusive for it may be more logical in many cases to consider that the contact, strictly speaking, is a localizing fault or at least a favorable structural form. The ground immediately adjacent to the intrusive is commonly greatly broken and often is a breccia reaching several hundred feet in width.

The localizing structural forms, in many if not most cases, will be obscured and even obliterated by the silication but careful mapping will reveal many which otherwise would not be noticed. The sulphide ore minerals and contemporaneous gangue are often localized in a striking way by post-silicate faults, shear zones, joints, and fissures which, in many cases, are easily recognized.

The sulphide and oxide ore bodies in tactite in contact zones are usually described as erratic, irregular, bunchy, and so on, but in many districts when carefully studied show more regularity than at first seemed the case. Many bodies are tabular for they are commonly related to favorable beds, fissures, or dike contacts. Horizontal to vertical, pod-like to chimney-like shapes are common. These are often related to fissure zones, intersections, and in a few cases to thrust faults.

## GENESIS

The paragenesis of these deposits does not seem to have been as generally and carefully studied as that of other types of deposits. Many students have considered the sulphides to be approximately or closely contemporaneous with the silicates. In the deposits of this type that I have mapped and/or studied with the microscope, including the Fierro-Hanover area, New Mexico; the Silver Bell, Christmas, Johnson Camp, Washington-Duquesne districts in Arizona; and Hedley district, British Columbia, the sulphides are later than and may replace the silicates and oxides. Knopf[6] says:

> In the earlier descriptions of the pyrometasomatic deposits the contemporaneous or simultaneous deposition of the silicates, oxides, and sulphides is frequently emphasized. In the more recent descriptions the tendency is to find an order of deposition. The most generally valid rule appears to be that formulated by Goldschmidt as a result of his studies of the Oslo region: Hematite is generally older than the silicates; magnetite and the iron sulphides are partly older and partly younger than the skarn silicates; and chalcopyrite, sphalerite, and galena are invariably younger than the silicates.

The most careful detail I have done on this type of deposit was on the Hanover, New Mexico, igneous contact.[7] This work indicated that in general the silicates were early. The sulphides were late and seem to have replaced the silicates. Large blocks of

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chapters 1 and 2.) I do not think most students of the western deposits would object to my statement. Bleaching and recrystallization of the limestone is considered a *metamorphic* change as well as the hardening of the shales.

andradite, as a rule not brecciated andradite, seem to have been replaced by the ore and contemporaneous gangue. In detail minerals with ferric iron were early whereas those with ferrous iron were late and came in just before and/or with the sulphides. B. S. Butler[a] says that the ferric and sulphide ions should be mutually exclusive in mineralizing fluids. There may be no known case where the sulphide and ferric iron minerals can be proved to have been contemporaneously and adjacently deposited. Gilbert[8] on an empirical basis makes the point that ferric oxide, particularly hematite, shows a spatial antipathy to high sulphide deposits.

As before noted, a number of students, including Barrell,[9] Goldschmidt,[10] Umpleby,[19] and Spencer,[18] have separated the early thermal or pyrometamorphism from the later pyrometasomatism. Behre, Osborn and Rainwater[11] at Calumet, Colorado, found distinct evidence of an earlier contact metamorphic group of silicates followed by deposition of ore minerals. Schmitt[12] did not find conclusive evidence at Hanover, New Mexico, that the thermal metamorphism, expressed largely by marble, porcellanite and ferro-tremolite, was early. The grain size of the late andradite, however, did match the grain size of the replaced marble beds. The thermal metamorphism, as a rule, though less striking, is many times more extensive than the thermal metasomatism.

### *Pyrometasomatic Deposits*

Of probable interest is the question of the relationship between the pyrometasomatic deposits in carbonate rocks and the quartz-sericite-pyrite-chalcopyrite deposits that often occur in a closely adjacent intrusive. The latter are usually classified as mesothermal, or if they contain such minerals as orthoclase, hematite, or epidote, as hypothermal. As Loughlin and Behre[13] point out, whether a deposit has the conventional features of the pyrometasomatic or the hypothermal may depend upon the character of the host rock, carbonate rocks giving one aspect, siliceous another. The problem is complicated by the fact that the tactite in some instances replaces (endomorphoses)[a] the intrusive. One will, perhaps, favor the hypothesis that usually closely adjacent mineralization regardless of facies will most commonly represent the same mineralization epoch. When andradite replaces the intrusive, there may be an influx of excess CaO from limestone, possibly from an underlying rock which is being garnetized and is giving off excess lime. Replacement of limestone by garnet seems to imply a loss of $CaCO_3$ (W. Lindgren from Butler[14]). Without an excess of $CaCO_3$, silica and iron may very well be deposited as quartz and iron oxide, or at a lower temperature and in the presence of sulphur, quartz and iron sulphide.

The classification of galena-tactite deposits as pyrometasomatic has been puzzling because of the commonly low temperature at which galena forms. The problem does not seem so troublesome now that we know that most sulphides are later than the tactite and probably form at a lower temperature.

Our knowledge of the genesis of these deposits in the field of physical chemistry is still on a very uncertain basis, to say the least, but is possibly better off than in the other classes of ore deposits. Recent ideas are strongly influenced by the fine work done on hot springs and fumaroles by the geochemists, largely stationed at Washington D. C. Since the high-temperature volcanic emanations seem always acid and gaseous it may be valid to assume that the early hot stage of metasomatism when silicates and oxides are largely stable is a time when a gaseous acid environment should prevail. The silicates and oxides would probably be stable in such a field but it is doubtful that they would be stable in a liquid acid or alkaline field.[7] The power of hot gases, particularly steam, to transport various elements has been emphasized in recent years, supported by experimentation. Of course, silica, iron, and the metals can be transported by the halogens, and if the vapor pressure of the halide is low, may be swept along in the current of steam.[15,a] The temperature of the early stage may approach a red heat. The pressure may be lower than the weight of a column of water to the surface, plus friction, because the gases may very well reach the surface largely unobstructed. The fumarolic and hot spring studies have shown that the "primary" gases usually are composed of predominating $H_2O$ and $CO_2$ and minor but influential amounts of $H_2S$, HCl, and HF. Small amounts of the metals and other characteristic elements such as As, B, N and C are usually present.

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[a] Oral communication.

[a] Should this be called pyroendomorphism?

The later stage when sulphides are deposited is doubtless at a lower temperature and could well be in a mildly acid to alkaline liquid phase, for the early silicates and magnetite apparently are in cases corroded and replaced. Thus Newhouse[16] in an apparently basic contribution, shows that the "primary" fluid inclusions in galena and sphalerite are predominately $H_2O$ with 12 to 25 pct of NaCl and some $CaCl_2$. There may be $CO_2$, but this could not be determined. Newhouse suggests that small amounts of $H_2S$ are probably often present as well as other elements. He further points out that qualitatively few of these elements diverge from those of the volcanic emanations, for the Na and Ca probably were picked up from the walls. The solutions seem to be chemically neutral. Other students have found abundant $H_2O$, $CO_2$, NaCl, and a few other elements in cavities in vein quartz. Just how these data fit into the picture is not clear. Qualitatively there is nothing antipathic to volcanic (igneous) emanations as a source for the fluids. The strength of the brine is perplexing. Needless to say, the whole matter of the character of the fluids needs further analysis, but encouraging progress seems to have been made.

The conventional assumption that the sulphide ores are deposited in an environment of an ascending column of liquid, largely water, from a deep source has been challenged by geophysicists who insist that there is no known force competent to raise this water. Further, in the hot spring fumarolic areas it has been shown[15] that characteristic volcanic gaseous emanations predominate below a relatively shallow shell of meteoric water. The geologists have based their argument on the belief that liquid water was necessary to transport silica and other material. As pointed out by Allen and Day[15] Elie de Beaumont noted that silica could be transported by fluorine in the gaseous state. This idea has not been given much weight for it is argued that the halogens could not usually have been very abundant because there are seldom important amounts of them in the minerals of pyrometasomatic deposits. Lately it has been suggested that many of the halogen minerals were not likely to be stable in most high temperature metasomatic environments anyhow, and the halogens probably passed through the contact zone to outer zones.

Perhaps we should not be too sure that meteoric water does not play some role in the mineralization, at least during the sulphide epoch. Of course, this has a dreadful sound to many. As has been pointed out by Lindgren, however, the time of mineralization is when dynamic disturbance, faulting, and so on, should be most active.

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[a] Grout comments here: "We have here at Minnesota found free transport of silica without any chlorine, fluorine . . . ."

It is my impression that other investigators have obtained similar results.

Surface waters should penetrate as deeply as there is an opening of moderate size and continuity and could return by artesian circulation boosted by gas lift. The fact that deep mines are dry may not be a valid argument against deeply penetrating meteoric water at the time of mineralization. Then, too, we may err in applying the theory of uniformitarianism to the problem. The activity, that is, the rate and magnitude, of faulting probably was greater, and rainfall may have been generally greater during the relatively restricted periods of great igneous-volcanic activity than during intermediate periods of which the present may be one.

Further, since the contact pyrometasomatic deposits might be *much* later than the adjacent intrusives and erosion is especially active in volcanic areas, erosion of some magnitude might ensue between the time of igneous intrusion and the sulphide mineralization. This, at a critical time, might decrease the distance between the meteoric water shell and the zone of mineralization. The distance would not normally increase by additional volcanism at this time, because the metasomatism usually occurs toward the end of the volcanic surge. The Yellowstone Park studies indicate that the fumarolic-hot spring epochs may cover long periods of time.[15] Some of the spring deposits at Yellowstone are covered by Pleistocene drift, and apparently the activity dates back at least as early as the time of the Pliocene rhyolite flows. There may have been time for important erosion.

### *Gain and Loss of Elements*

A discussion of the elements gained and lost in the contact pyrometasomatic zones may be of interest. Many investigators have shown that the zones gain large amounts of substance. Lindgren[17] showed that the metasomatic rock at Morenci gained a ton net of substance per cubic meter and Schmitt[7] reached similar conclusions. His work at Hanover indicated a gain of two tons and a loss of one ton of substance per cubic meter. The large amount of energy which must have been expended to make such changes is worthy of note. The limestone largely gained silica, iron, zinc, and sulphur and lost some lime and carbon dioxide. The aluminous rocks in general, shales and igneous rocks, lost silica and the alkalies and gained iron. When at Hanover the average composition of all the rocks of the zone before metasomatism is compared with the average composition, after metasomatism, the net gain is revealed to have been largely made up of iron, sulphur and the other metals with a moderate amount of silica. This silica may have come from the *alteration* of aluminuous rocks below the visible zone, probably from the intrusive itself. In other words, for this relatively small amount of exotic silica we may not need to look to the magma. If this can be accepted, then the material definitely added to the zone is composed only of elements usually found in the volcanic (igneous) emanations, i.e., Fe, S, and varying amounts of the other metals.

Probably the amount of exotic silica in "contact" zones has been exaggerated. The abundant porcellanites are usually called silicified shale or silicified limestone, implying that silica has been added when usually they probably are actually only baked shale or baked shaly limestone. The aluminous rocks lose silica when they are epidotized and chloritized. These metasomatisms, as a rule, prove to be later than the pegmatitic epoch.

### ECONOMICS, STUDY, MAPPING, DEVELOPMENT

For several decades it has been the common belief among many mining engineers and geologists that the usually called "contact metamorphic," actually pyrometasomatic ore deposits, are as a class usually economically marginal and/or small in size. This seems to have little factual basis. Thus in the Southwest such major mining districts as Bisbee and Clifton-Morenci in Arizona, and the Central district in New Mexico, have much of their ore tonnage in this class. Many moderately sized districts or mines can be named such as Silver Bell, Christmas, and Johnson Camp in Arizona. There are numerous others. Surely there are as many disappointing examples of ore mineralization in limestone in the other major classes as there are in contact ores; the alteration accompanying them is just not as spectacular, and, therefore, not likely to be so misleading.

Mining exploration and development of the ore bodies in the contact pyrometasomatic zones to be efficient must largely contemplate three factors: (1) those elements of theory which now seem to be on a firm basis, (2) the known local shape and distributive habits of the ore, (3) the general shape and distributive habits of ore in similar zones.

The elements of theory which are the most useful include: (1) The ore will normally be controlled by structural forms including small and large faults, joints, channels, or masses of breccia, and rarely folds. (2) Since the sulphides are usually later than the silicates, one must pay particular attention to post-silicate, pre-sulphide faults or other "ground preparation." (3) The "marble line," i.e., the contact of the garnet with the unreplaced marble, is a zone especially favored by ore, but, nevertheless, the so-called "dead-garnet" masses close to the intrusive, when broken, can be a host for ore. (4) Certain beds, particularly the purer limestone beds, *even after they have been replaced by silicates*, are as a rule the most sensitive to replacement; such beds just below thick shales are especially favored. (5) The contacts of dikes are places favored by ore particularly when cut and broken by post-dike pre-ore movement. (6) The actual contact of the "main" intrusive may or may not be a zone of localization of ore.

From a more empirical standpoint, knowledge of the local habit and distribution of the ore bodies and the general habit and distribution of this type of ore is important, especially when the mine development has not been far advanced.

The theoretical and more empirical data must be measured and/or noted and expressed on maps of a scale adequately large. The microscope is essential in working out paragenetic relationships which when made clear have aided in the discovery of ore.

In many districts the ore bodies will show a pattern of preferred orientation. Efficient development work will take advantage of this. If a preferred orientation is pronounced, a few well selected cross-sections when adequately drilled will intersect many of the ore bodies. Diamond drilling has proved to be especially effective in exploring this type of deposit, because usually the limestone, silicates, magnetite, and sulphides give high core recovery.

The careful mapping and study of the gossans in the tactite area may be especially fruitful. In one district when a boundary was drawn around the scattered small patches of gossan on the surface, it was found that the area outlined very closely reflected the outer boundary of the ore bodies which were up to 600 ft below.

## Conclusions

1. The term contact metamorphism should be limited to its original meaning, i.e., change of character without substantial addition of substance in the zones adjacent to igneous intrusives, and Lindgren's term contact pyrometasomatism should be used for changes at elevated temperature at a contact with substantial addition of substance. The term tactite seems to be the

most useful general term for pyrometasomatized rock.

2. The control of silicate and ore mineralization in contact zones by structural forms gains more and more emphasis as detailed mapping is advanced in the various districts.

3. The generally early age of the silicates and late age of the sulphides is well established. The work of the geochemists has renewed emphasis on the formation of the silicates by pneumatolysis. Final proof may not be far off that the normal gaseous volcanic (igneous) emanations supplied the heat and substance to form the early silicates. This writer believes that most of the silica added to the zone comes from local alterations (metasomatisms) and that we need not depend on the magma for the silica. The chief substances then, normally added to the aureole may be iron, sulphur, other metals, and rarely As, Cl, F, and B. During the metasomatism usually nearly all the Na, K, and $CO_2$ and some of the CaO is lost from whatever rock is altered that contains them. $Al_2O_3$ and MgO are relatively stable.

4. The work of Newhouse[16] suggests some universal, possibly simple, chemical field in which the sulphides are deposited. This field may well be the same regardless of the classification of the deposit. The simple but strong brine with minor $CaCl_2$, $CO_2$(?), and $H_2S$ may be a composition reached by the mineralizing fluids after they have pyrometasomatized and sericitized deeper zones. Newhouse's results imply a fluid which, since it is probably below the critical temperature of water, would be a liquid phase. This writer ventures the suggestion that some of the water may have been meteoric.

5. The widespread belief that as a class contact pyrometasomatic deposits are economically marginal and/or small in size seems to have little factual basis. There are as many disappointing examples of mineralization in limestone in the other major classes as in contact ores, but the alteration accompanying them is not so spectacular, indeed often is largely absent, and, therefore, is not likely to be so misleading.

6. The exploration for ore in the contact pyrometasomatic zones to be efficient must usually be preceded by large scale detailed mapping and geological study. The common circumstance that the ore bodies have a preferred orientation is an excellent guide. Diamond drilling is extensively used because of the facility with which cores are recovered from tactite.

## ACKNOWLEDGMENTS

I am greatly indebted to Drs. A. Knopf, F. F. Grout, B. S. Butler, and T. O'Neill for their careful and objective criticism of this paper.

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# The Appraisal of Ore Expectancies

By Edward F. Fitzhugh, Jr.,* Member A.I.M.E.

(Chicago Meeting, February 1946)

## Abstract

Quantitative appraisal of the chances of finding various tonnages and grades of ore clarifies any exploratory development proposal. Ways are discussed of appraising chances in conformance with probability principles, and graphs are suggested to show tonnages, grades and the corresponding judged expectancies.

## Introduction

When exploratory development of a mineral deposit is contemplated, three interrelated questions normally must be answered before a sound decision can be reached. How much will the program cost? How much will the anticipated ore add to the immediate value of the mine? And what is the chance of developing this ore? To justify development, the deposit should present a favorable disparity between the three factors, cost, chance and return. If, for example, there is a 50-50 chance of success, the ore that is sought should have an immediate worth† (after deducting all subsequent charges that it must bear) of more than twice the cost of the development. Likewise, if the chance of finding the ore is but one in five, that ore should enhance the immediate value of the mine by more than five times the cost of developing it. With sound economic and operating assumptions, the cost of exploratory development and a present value of anticipated ore may be estimated within reasonable limits. Less general attention has been given to appraising ore chances in comparable terms. The exploratory experience of the mining industry is broad enough that knowledge, vision and judgment may be combined to yield more tangible evaluations of ore expectancy than are now customary. A way of making such appraisals is suggested herein.

The mathematics of probabilities cannot be applied directly to the search for ore. By coupling applicable principles of probability, however, with pertinent experience gained in mineral exploration, an estimate often can be made of the chance of finding a certain tonnage and grade of ore in a specific location. Such estimated, or judged, expectancies for the several blocks of ground in a deposit may be combined in a simple chart to show the relationship of the various exploratory possibilities and the potential magnitude of the deposit.

An engineer or geologist may find that the quantitative thinking required for an appraisal of this type is an aid in clarifying and coordinating his views and recommendations. For a person wishing to understand these views, an ore-expectancy

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Published by permission of the Director, Bureau of Mines, U. S. Department of the Interior. Manuscript received at the office of the Institute Feb. 15, 1946. Issued as TP 2090 in Mining Technology, January 1947.

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† Interest and deferment allowances are proper elements in comparing cost and return. Rather than calculate a present value of ore that has not yet been explored and developed, it may be preferable to adopt the end of the development job as the date of comparison. Then, interest that money spent for development would otherwise earn by this date is (for this purpose) a development charge; the cost is weighed against the amount that would be a reasonable cash purchase price of the ore that is sought, payable at the close of development and in the circumstances that will presumably exist then.

chart is a clear, concise summary of conclusions.

As obvious good bets for exploratory development become scarcer, and as ventures with smaller disparities between cost, chance and return arouse interest, the need for more precise evaluations will increase.

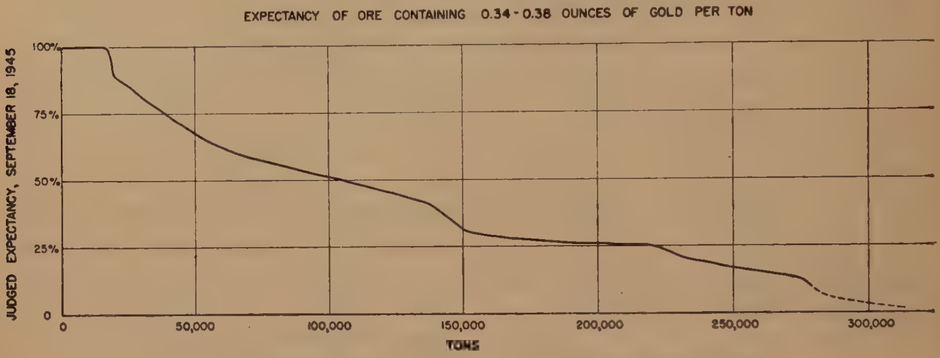
EXPECTANCY OF ORE CONTAINING 0.34-0.38 OUNCES OF GOLD PER TON

FIG. 1.—EXPECTANCY OF ORE IN A PARTLY DEVELOPED MINE.

### TONNAGE, GRADE AND PROBABILITY

All deposits have limits. So, when we contemplate the possibility of proving new ore in a deposit and envision progressively greater tonnages, the corresponding probability decreases. The judged probability may be plotted against tonnage to yield a comprehensive and readily understood graph. Fig. 1 is an expectancy chart for a mine having 16,000 tons of proved ore and for which there are believed to be better than 50-50 chances of an additional 87,000 tons. Smaller chances are judged to exist for further tonnages as shown.

As this chart is constructed, the scaled tonnage represents the cumulative sum of the blocks of proved and potential ore under consideration. The curve is obtained by arranging these in order of declining expectancy, without regard to the position of the ore blocks in the ground. For a development program to have an expectancy of, say, 60 per cent or better, the work must be devoted to the corresponding portions of the deposit. If a specific tonnage is desired, the position of the curve shows the apparent chance of developing this amount of ore as the probabilities are judged at the time the appraisal is made.

On a chart of this type, the small and unpromising prospect is represented by a hyperbolic curve; large reserves of proved and nearly proved ore maintain the curve at high levels for suitably scaled distances to the right of the ordinate.

### *Basis for Appraisal*

The basis for an expectancy appraisal encompasses all that is known about the deposit through inspection, sampling and measurement of available exposures, together with whatever may be inferred from the character and structure of the ore and the surrounding rocks, and from comparison with neighboring mines and with similar occurrences in other districts. For ore to be classed in the higher ranges of assurance, the extent of development must make for a high ratio of facts to inferences.

Attempts to arrive at a formula relating the ore expectancy to the progress of development have not been widely successful. The more fully an ore body is exposed, the more certain we may be of its presence and of the tonnage and grade. Except in a few districts, however, the significance of physically comparable partial development is seldom the same at two mines. Just because full assurance is reached at most mines when ore is blocked out and sampled on four sides in units of appropriate size, it certainly does not follow that three exposed sides make for a 75 per cent expectancy, two sides for a 50-50 chance and one side for one chance in four. In the majority of cases, the expectancy of ore that can be seen on three sides or on two opposite sides is

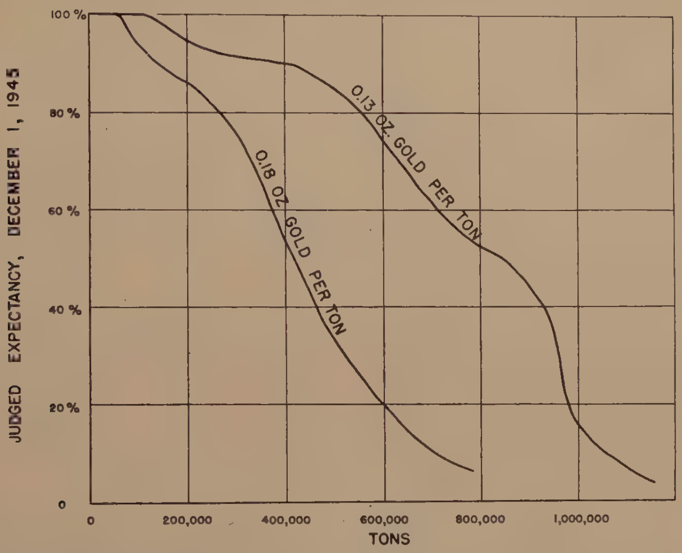

FIG. 2.—CURVES WHEN MORE THAN ONE MINING PROCEDURE IS FEASIBLE.

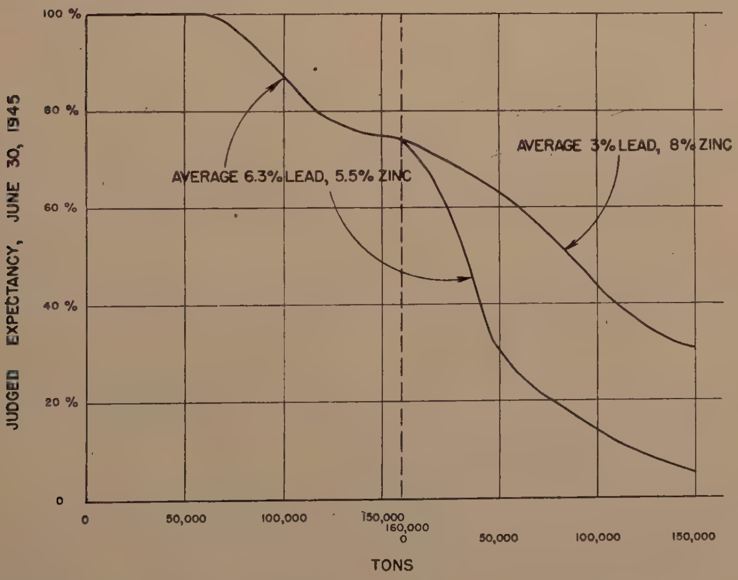

FIG. 3.—CURVES WHEN A MAJOR CHANGE IN GRADE IS EXPECTED.

relatively high, while the chance of continued mineralization beneath a bottom-level drift or beyond a terminal raise is a geological rather than a mathematical question.

### *Grade Curves*

The grade features, whether calculated or envisioned, that can be entered effectively on a tonnage expectancy chart are limited to broad averages. Where more than one mining procedure is feasible, and separate cutoff and average grades are to be considered, two or more curves may be plotted on the chart, as in Fig. 2, or separate charts may be drawn. If a major change in grade appears likely, a bifurcated line, Fig. 3, may show this. Here zinc is expected to increase at the expense of lead in the less known extensions of the deposit. For clarity, it is well to start the tonnage scale anew at the point where the grade curves diverge. In the example shown, the judged expectancy to the right of the second ordinate is twice as good for 50,000 tons and three times as good for 100,000 tons of ore richer in zinc as it is for ore comparable to the known reserve.

### QUANTITATIVE APPRAISAL AND CHART CONSTRUCTION

To calculate an ore expectancy mathematically, we would have to be able to measure the influence that each pertinent factor exerts on the probability, weigh these factors precisely, and know the exact interdependence of the several factors. In a firm mathematical sense this is impossible. By drawing on experience, however, and estimating the influence, weight and interdependence of the known pertinent factors, both favorable and unfavorable, a judged expectancy may be reached.

A practical way of working out such a quantitative estimate is first to limit the block of ground under consideration. Blocks may be either mining units or geological units or both. Starting with the block or blocks that are best known, appraisals of the chances are made. The chances for possible extensions are related to these initial estimates.

This procedure is illustrated by the following example. Fig. 4 is a plan and longitudinal elevation of the workings on a vein which dips steeply to the west in homogeneous, flat volcanic flows. The contact between the igneous rock and the underlying shale, as projected from the surface, is shown on the elevation. The vein structure is strong, the mineralization is reasonably uniform, and the surface exposures of the two ore shoots are quite similar. The surface trenches have not exposed the south end of the southern shoot. Since the appraisal is a matter of individual interpretation of what can be seen, measured and sampled, and interpretations are seldom identical, the geological and sampling details are omitted from this illustration of procedure.

The calculated tonnage in the well-developed block *A* is 19,000 tons. The type of ground and number, spacing and variations of samples and measurements make it highly improbable that stope production would vary from this by as much as 15 per cent (a judged figure). So in the tabulation of estimated expectancies 100 per cent is assigned to 16,000 tons of which we may be "positive," 90 per cent, or nine chances in ten (a judged figure), to the 3000 tons difference, and a corresponding 10 per cent to another 3000 tons for possible excess ore.

The lower limit of block *B* is at an elevation that is suitable for a deeper adit level. Estimation of tonnage and corresponding chance is made after weighing the favorable and unfavorable factors. In this instance, there is no reason to expect uncompensated pinching or swelling, so the tonnage is calculated by projecting the ore area down the rake from the bottom of block *A*. To arrive at a probability figure, it is often helpful to start with maxima and minima, then, balancing the pros and cons, narrow the range to a point where splitting the difference is of small consequence. For the calculated 32,000 tons in block *B* the chance is judged to be better than 50-50 but less than five to one. Reviewing and weighing the factors, two chances in three are finally assigned to the block. The greater hope for a lesser tonnage in this ground is shown when the points on the chart are connected by an appropriately smoothed curve.

EXPECTANCY ESTIMATE

| BLOCK | TONS | EXPECTANCY |
|---|---|---|
| A | 16,000 | 100 % |
| | 3,000 | 90 |
| | 3,000 | 10 |
| B | 32,000 | 67 |
| C | 59,000 | 44 |
| D | 25,000 | 25 |
| E | 15,000 | 60 |
| F | 15,000 | 30 |
| G | 13,000 | 40 |
| H | 13,000 | 20 |
| I | 42,000 | 26 |
| J | 42,000 | 13 |

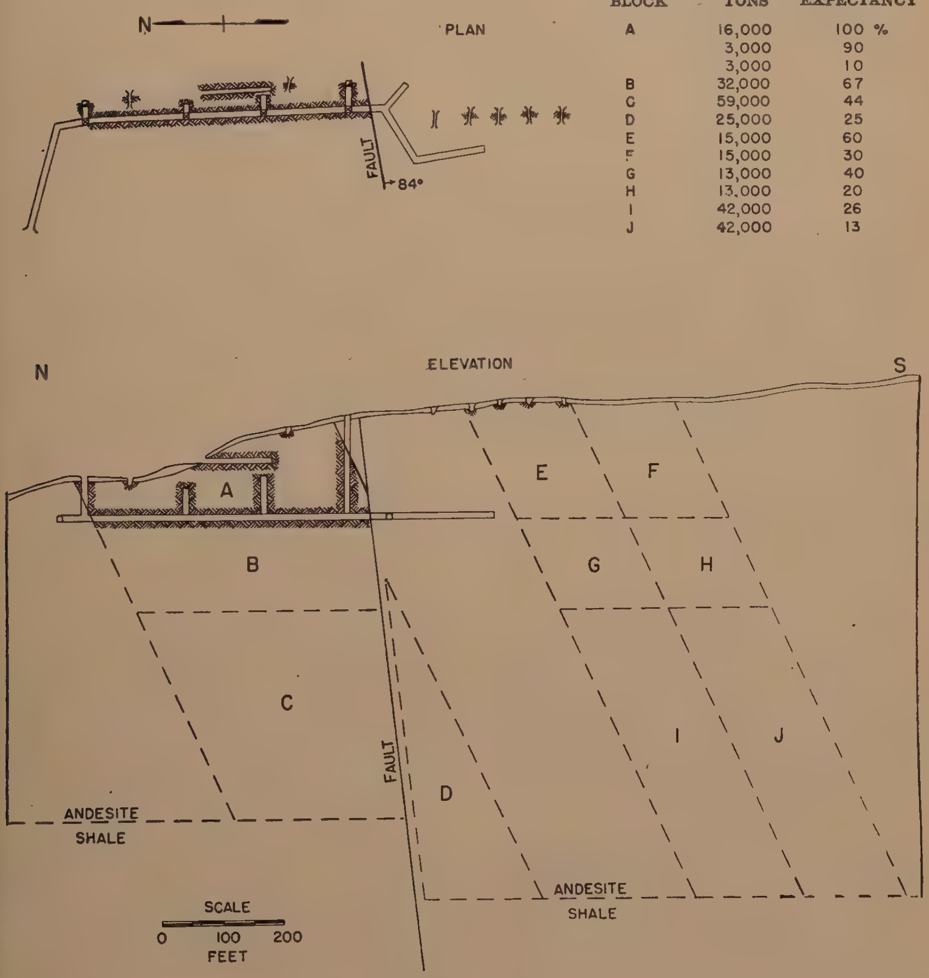

FIGURE - 4

FIG. 4.—PLAN AND LONGITUDINAL ELEVATION OF WORKINGS ON VEIN ILLUSTRATING EXPECTANCY APPRAISAL PROCEDURE.

A lesser chance might now be arbitrarily assigned to block *C*, but the

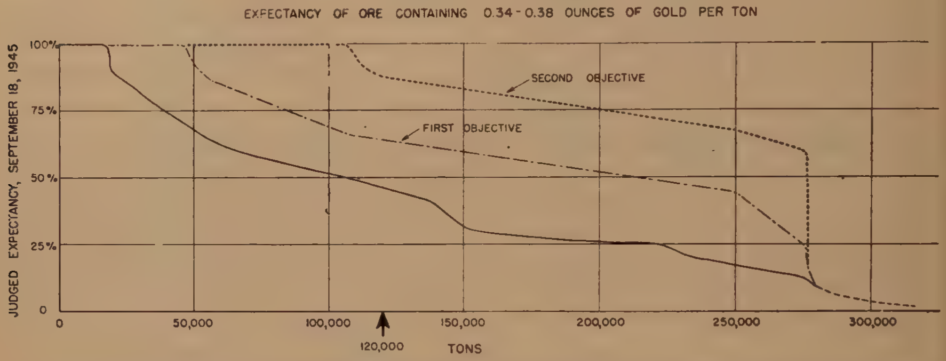

FIG. 5.—OBJECTIVES OF TWO SUCCESSIVE STAGES OF DEVELOPMENT PROGRAM PROPOSED FOR MINE IN FIGURE 4.

mathematical relationship of probabilities suggests a better way. Applying a bit of imagination, one can envision a drift along the bottom of block *B* and raises through the block. Then, assume that the ore in the drift and raises bears out, essentially, the favorable hope to which an expectancy of 67 per cent is assigned right now; that is, assume that the block has been developed and that the results have been satisfactory in the way that was anticipated. With this imagined background, weigh the pertinent factors and assign an expectancy to block *C*. By making the appraisal for *C* on the assumption that the ore has been proved in block *B*, we have a compound probability; the probability for *C*, as determined, should be multiplied by the probability of *B* to obtain the present expectancy of the greater tonnage in *B* and *C* combined. In this case, the contingent ore probability of *C* was estimated at two chances in three, making the present expectancy four chances in nine for increasing the tonnage by the calculated 59,000 tons in block *C*.

In treating compound probabilities, it should be noted that we are dealing with ore expectancies rather than with what is most likely. So, even if the chance for the estimated tonnage in *B* was judged to be small, we should assume that this chance was fulfilled when the contingent tonnage and chance for *C* are appraised.

Comparing the hopes for block *D* with the chances north of the fault, and allowing for the uncertainties introduced by the faulting, the chance of 25,000 tons is estimated at one in four.

For the south ore shoot there is direct evidence favoring the extension of visible ore from the surface to the drift level with a rake parallel to the north shoot, as shown in block *E*. The chance for 15,000 tons in this block is judged, after due reflection, to be three in five. Estimation of the compound chances as the deeper increments, *G* and *I*, are added parallels the appraisal of the north ore body.

More sheer guessing is involved in projecting ore to the south of block *E* than in any other step. This shoot may be about as long as the north one, and a series of possible lengths with corresponding chances might be considered. In this instance a 50-50 chance of an ore extension equal to that already exposed appears reasonable. With this assumption, the probability of block *F* is half that of *E*, or three in ten; the possibilities of deeper extensions are correspondingly less.

Plotting the tonnages and chances in the table of estimated expectancies yields the curve in Fig. 1.

### APPLICATIONS OF THE CHART

In addition to summarizing the present outlook for a deposit, the chart may be used to show possible future expectancy positions. Fig. 5 shows the objectives of two successive stages of a development program proposed for the mine in Fig. 4. Likewise, appropriate future expectancy curves facilitate the comparison of different ways of exploring a deposit.

A minimum amount of ore needed for a successful mining operation may be emphasized on the tonnage scale. For a range of economic assumptions, the tonnages needed under varying conditions may be set up as a grid or in other convenient graphic form to parallel the tonnage scale.

Quantitative appraisals of the ore expectancies afford a common basis for comparing dissimilar deposits.

As exploratory development progresses, expectancies change. When a deposit is being opened up, revisions of the appraisal at significant stages will focus attention on the advisability of continuing the development, starting mill construction, or quitting the venture. Similar periodic appraisals for producing mines show how effectively development and exploration may be keeping ahead of production.

Exact agreement between two mine examiners is not to be expected. The chart itself is an approximation. Furthermore, differences of experience, viewpoint and personal factors make for divergence in judgment. The limited trials that have been made with this type of appraisal suggest that similar over-all views make for quite comparable curves. When a significant difference of opinion occurs, its economic implications are apparent.

The engineer or geologist *guarantees* only the ore to which he assigns an expectancy of 100 per cent. His estimate of grades, tonnages and chances for the remainder of a deposit is, like any estimate, the best guess that he can make after weighing all available information. Recognition that adverse possibilities exist and that no blame might attach to the appraiser if ore rated at less than 100 per cent expectancy were not found, however, does not extend a license for unfounded optimism or for unjustifiable conclusions.

### CONCLUSION

As easily recognized, favorable exploratory opportunities become scarce, the need increases for quantitative information on the chances involved in developing ore. While our knowledge is by no means complete, in many instances reasonably close estimates of ore chances may be made. It is suggested that appraisals of the character described herein are suited to a number of practical, working purposes.

# A Prospecting System Developed at Balatoc Mine

By Paul A. Schafer,[*] Member A.I.M.E.

(New York Meeting, February 1942)

Balatoc mine, the principal gold-producing mine in the Philippines, is 10 miles southeast of Baguio, in the Cordillera Central of northern Luzon.[1] The region is a deeply dissected plateau, ranging in altitude from 3000 to 5000 ft. above sea level.

The rocks consist chiefly of granodiorite, diorite, and remnants of metamorphosed sediments, cut by dikes of andesite porphyry and quartz-diorite porphyry. All are intruded by a volcanic breccia plug, oval in horizontal section, about 3200 ft. long and 2300 ft. wide, which is the north central part of the mineralized area.

Three main vein systems have been developed (Fig. 1): 96, 231 and 364, which strike about N.70°E. and dip 50° to 90°S. Each system is a braided zone consisting of many individual components. They are disturbed by a series of premineral faults or shear zones striking about N.45°E. Twelve of these produce important effects on the persistence, character, and quality of veins. From west to east the faults are named (after claims): Hog, Big Gun, Bull, Pine, Lolita, Bar No. 1, Bar No. 2, Curt, Sun, Helen, Itoe, and Frog. They are early breaks formed prior to the N.70°E. vein fractures. Mineralization occurs on them but, with several prominent exceptions, they are too low grade to mine.

A number of northwesterly striking faults complete the structural pattern. Although they are relatively abundant, the effect on the veins is so small that they do not present any problems in mining or development. All of these faults that have been observed are later than the N.70°E. veins and offset them to the left.

## Northeast Faults

The early northeast faults[*] are notably continuous in spite of their small displacements, 20 to 100 ft. Frequently they appear so weak that their importance is masked and their identity clouded. The gouge along any of these faults may vary from exceedingly weak at one point to very strong at another point only a short distance away. In other words, the amount of gouge depends on the changes of attitude of the fault plane, and is no indication of the extent of the movement. Places where walls are spread apart are filled with gouge squeezed from tight places. Also, mineralization sometimes obscures fault gouges by replacement.

The mineralization of northeast faults has produced several important ore bodies; namely, 231 west on Pine fault and 96 vein on the same fault. To a smaller extent, Lolita, Curt, Bull, Bar No. 1, and Sun

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Manuscript received at the office of the Institute Aug. 20, 1941. Published by permission of Mr. A. F. Duggleby, Vice President, Balatoc Mining Co. Issued as Contribution 130, January 1943.

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[1] Andrew Leith gives a map of the district, which shows the relative positions of the Balatoc, Itogon, Antamok and Big Wedge mines [The Geology of the Baguio Gold District. Dept. Agr. and Commerce, Commonwealth of the Philippines, *Tech. Bull.* No. 9 (1938)].

[*] The distinction of certain premineral faults was made before 1936 by A. F. Duggleby and G. C. Worthington, who took some account of this structural relationship in development.

faults are productive, but for the most part northeast faults are low grade or barren.

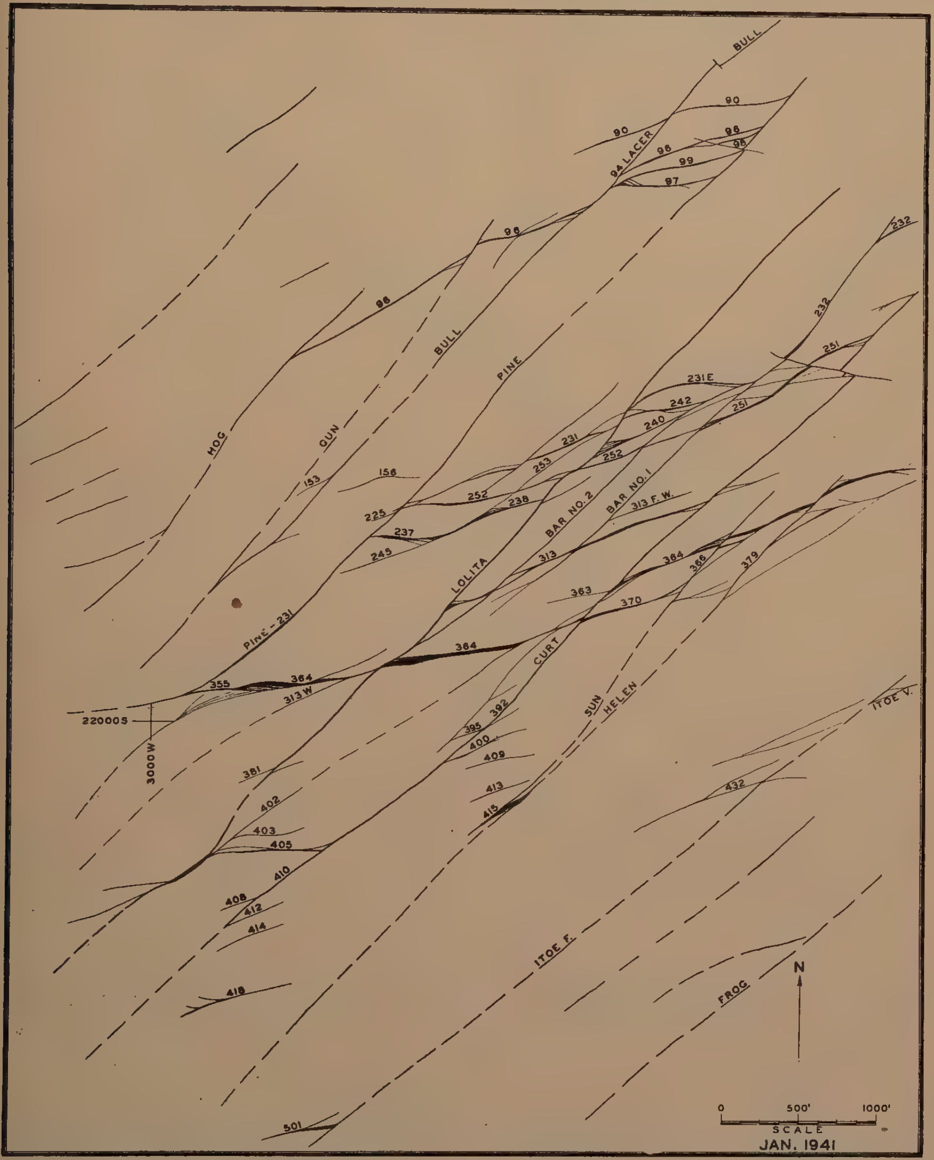

FIG. 1.—BALATOC VEIN PATTERN. FAULTS ARE NAMED AFTER CLAIMS; VEINS ARE DESIGNATED BY NUMBERS.

## VEINS

The veins vary in strike from east-west to N.60°E. and dip from 50°S. to vertical. They are arranged in three principal parallel groups spaced roughly 1000 ft. apart. The major vein systems 96, 231 and 364 are continuous for 2000 to 5000 ft., but some veins exist only in the block between one northeast fault and the next. For example, 97 vein lies between Bull fault and Pine fault; the 240 vein occurs only between Bar No. 2 and Lolita faults; the 313 vein is important between Lolita and Curt faults, and although its structure continues beyond Curt fault, it has not been found beyond Sun fault. Vein 231 terminates westward on Pine fault and eastward on Sun fault; 96 vein fails to pass through Pine fault on the east, and Hog fault on the west. Thus it may be concluded that a vein may not pass beyond a northeast fault, and, in fact, it may be strictly limited between two faults. This fact has an important bearing on development, as will be shown later.

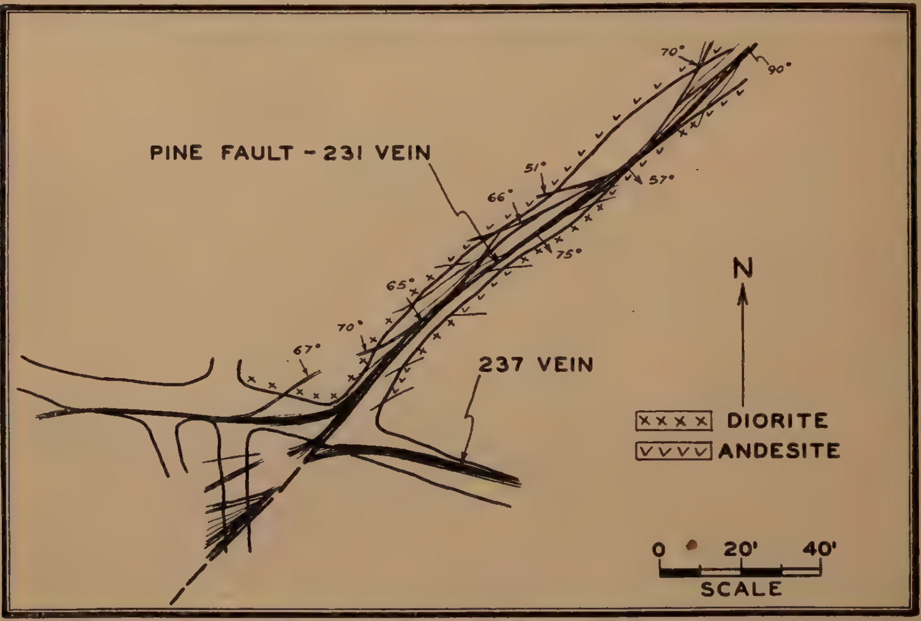

FIG. 2.—JUNCTION OF 237 VEIN WITH PINE FAULT.
The vein turns parallel to the fault and pinches rapidly as it approaches the fault.

## FAULT AND VEIN JUNCTIONS

Two principal types of fault and vein junctions occur. First, the vein approaching a fault may turn more nearly parallel to the fault, pinching as it does so (Fig. 2). This indicates that active shear in the vicinity of the fault has influenced the vein. Often a drift along a fault does not disclose the presence of the vein junction.

In the second type of junction, the vein begins to split into numerous branches as it approaches the fault (Fig. 3). In general, the vein turns toward the fault to approach it at a wide angle. The junction of 251 vein and Bar No. 1 fault illustrates this type (Fig. 4). If the vein continues on the other side of the fault, it follows the fault for an unpredictable distance, breaking away on the other side; in fact, there is no assurance that it will continue beyond the fault at all. Usually the part along the fault is tight and barren. Thus a crosscut frequently passes through the barren fault without disclosing a vein that may lie as close as 20 ft. in either wall (Fig. 5). Such accidents of crosscutting have happened in many places at Balatoc: On the 1500-ft. level, 18 crosscut (Fig. 6), and 0 crosscut pass through the Lolita fault and Bar No. 1 fault, segments respectively of 313 vein; and on the 1700-ft. level 0 crosscut went through the tight, barren Bar No. 2 fault with ore only a few feet away in both walls.

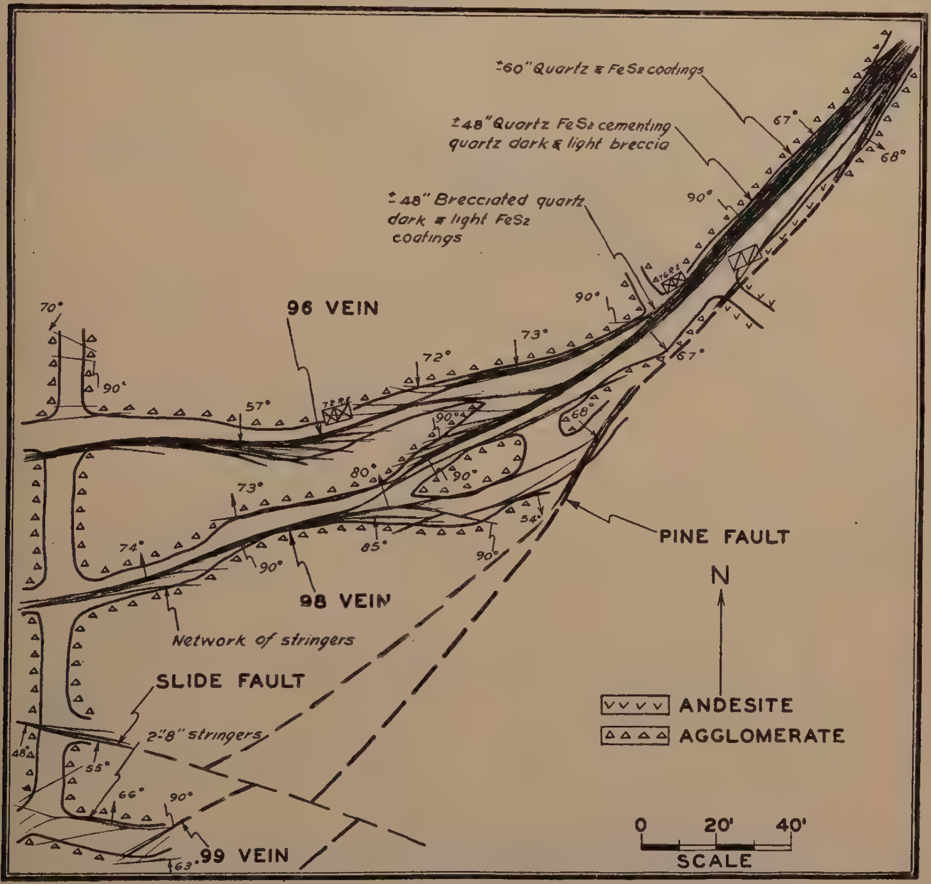

FIG. 3.—CONFLUENCE OF 96, 98 AND 99 VEINS INTO PINE FAULT.

### PROSPECTING PLAN

The foregoing description shows that it is of fundamental importance to identify all important northeast faults and to lay out work on the basis of their arrangement. Certain rules must determine any plan for exploration, as follows:

1. Drifting along a fault does not prospect the blocks adjacent to that fault because veins may approach the fault in such a pinched or scattered condition as to be unrecognizable or appear to be unimportant.

2. Any crosscut explores only the fault block within which it lies, for a productive vein existent in an adjacent fault block may not enter the block explored by the crosscut.

In order to apply these two rules to the best advantage, a prospecting plan was devised by the writer in the fall of 1937, which showed in detail the blocks completely explored and those which cannot be considered as explored (Fig. 7). The laying out of future work was controlled in large measure by this scheme.

Boundaries of blocks (north and south) are east-west lines representing the extreme strike of veins (shown by 231E, 97, 252W,

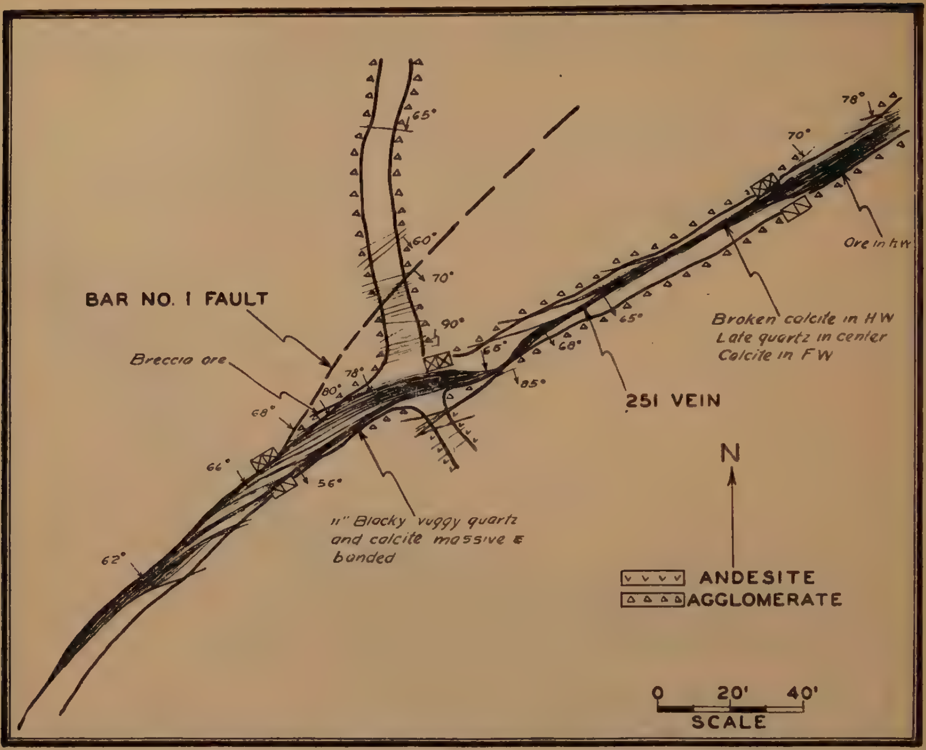

FIG. 4.—PLAN OF JUNCTION OF 251 VEIN AND BAR NO. 1 FAULT. THE ORE SPREADS OUT FANWISE TOWARD THE FAULT.

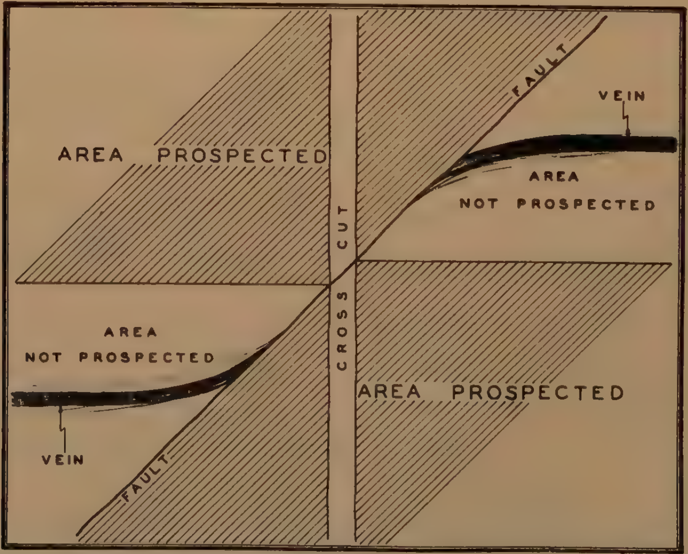

FIG. 5.—IDEALIZED SKETCH SHOWING AREAS PROSPECTED BY A CROSSCUT THAT PASSED THROUGH A PREMINERAL FAULT.

355). Faults and veins indicated by pertinent data were projected 200 ft. from the 1300-ft. and 1700-ft. levels to be included with those mapped by data from the 1500-ft. level. The 1500-ft. level was taken as the basis for the map (Fig. 7) because development is most complete at that level.

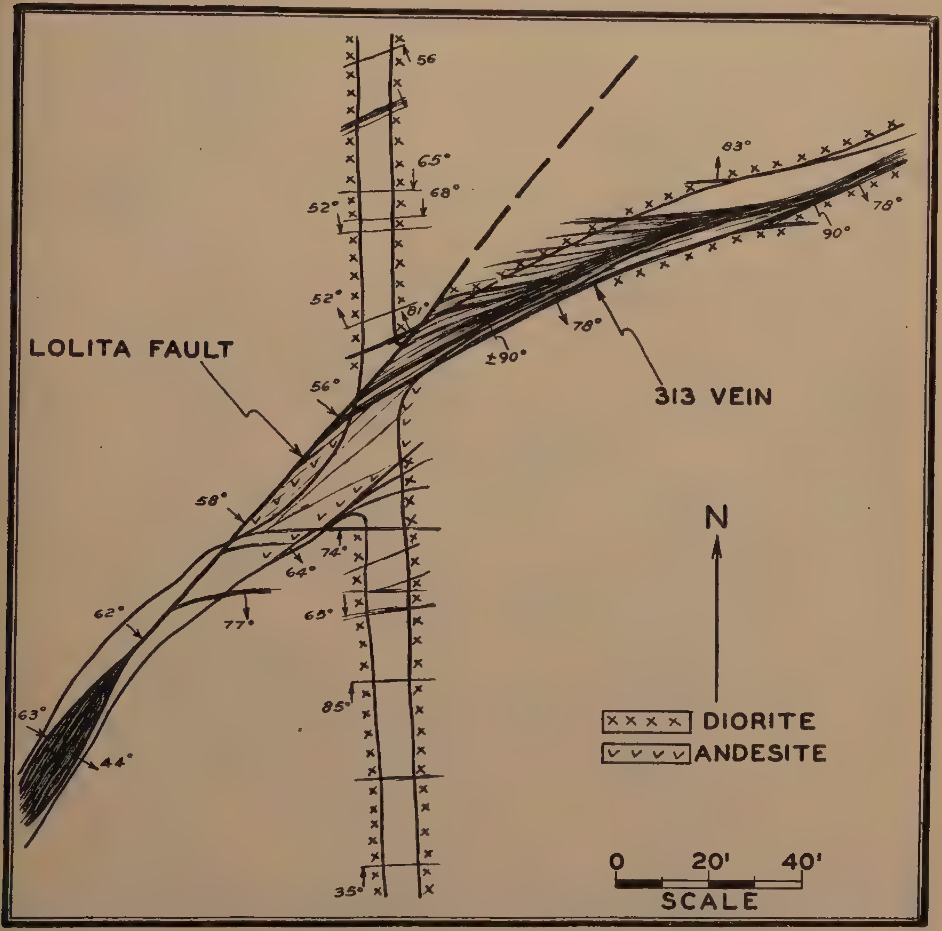

Fig. 6.—Development crosscut nearly misses 313 vein where it is pinched on Lolita fault. For years it was not known that fine ore was less than 20 feet from the crosscut.

Each year since 1937 the prospecting map has been brought up to date to show the areas prospected during the year and the unprospected areas remaining. It is not intended to eliminate the unprospected blocks rapidly, but to explore them gradually in the most economical manner compatible with good mining; for example, to satisfy the need for fill in the cut-and-fill stopes.

The advantages of the plan are apparent. Prospecting work is not duplicated. Each block is thus adequately explored in the cheapest manner. It is definitely ascertained when an area is completely prospected. Crosscuts may be laid out in the most efficient manner to cover a block thoroughly. The system has delimited many promising areas in regions formerly thought to have been adequately covered by crosscuts (for example, the ground between 0 X-cut and 90 X-cut, Fig. 7).

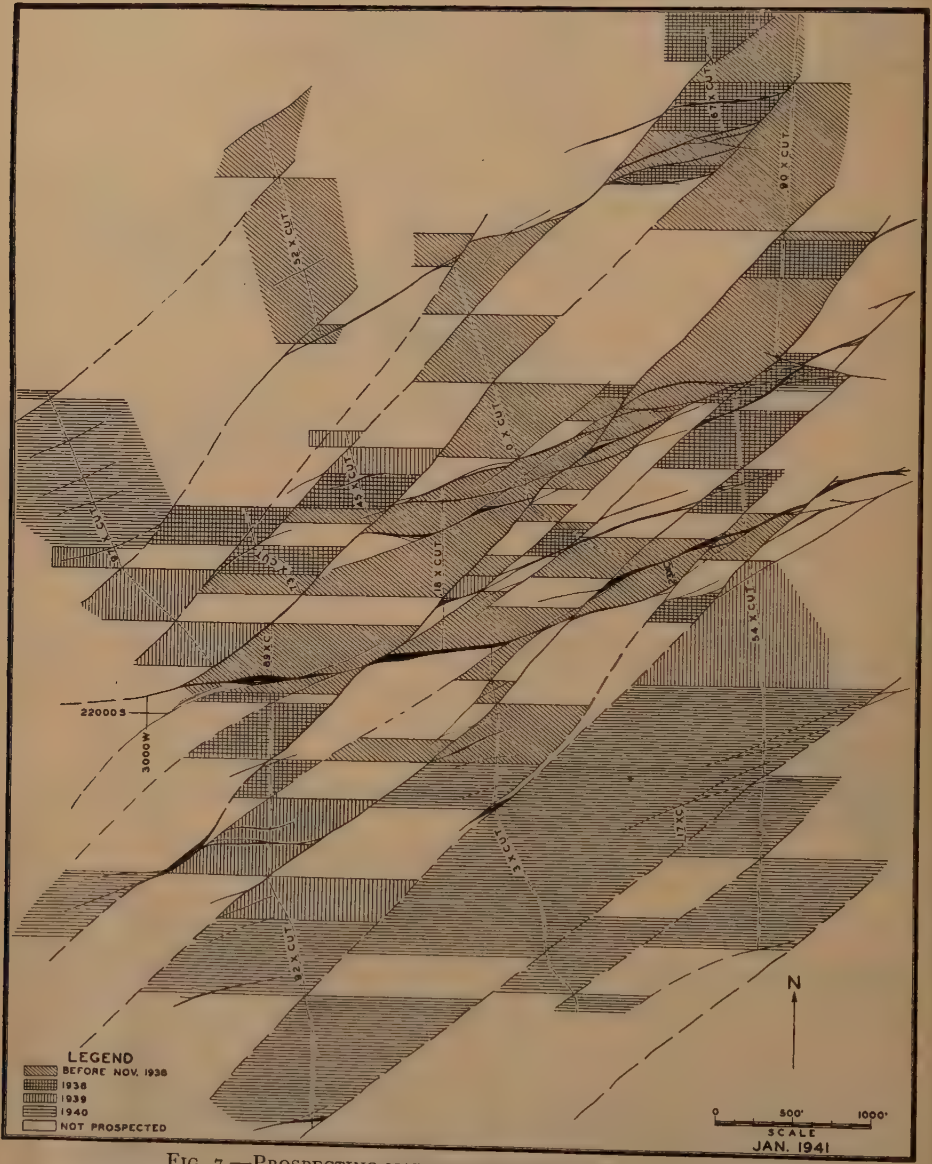

FIG. 7.—PROSPECTING MAP, BASED ON THE 1500-FOOT LEVEL.
Shaded areas are considered prospected, the pattern of shading indicating the year that prospecting was accomplished. Unshaded areas are not prospected.

## RESULTS

It has been possible to note the results of the application of the prospecting plan for 3½ years. During that time numerous blocks have been partly or wholly covered by development crosscuts. The veins exposed by this work are 153, 156, 97, 251, 191, 363 (Figs. 1 and 7). Of these 97, 251, and 363 are known to be productive the others are at present considered nonproductive. In addition, work in virgin country south of 364 has discovered several productive veins (403, 405, 410, 414, 412, 415, 409, 418, and 432).

Considering the disposition of shear structures, it was considered expedient to drive three crosscuts south into new country, to achieve as thorough coverage as possible. Other factors also influenced the decision to drive three crosscuts instead of one. Itogon mine, east of Balatoc, discovered a highly productive vein—the Itoe vein, which projects into Balatoc south of the 364 vein a short distance from the Balatoc boundary line. Crosscut 54 was driven near the boundary to cut this structure. Significantly, the Itoe failed to extend beyond a fault (Balatoc calls it the Itoe fault), and did not reach Balatoc ground. Consequently 54 crosscut failed to disclose ore, except a small shoot on 432 vein south of the Itoe projection. Crosscut No. 3 (1300-ft. level) cut two promising structures, each with some proved ore—veins 409 and 415. Crosscut 92 encountered two important vein zones—the 400 vein zone (402, 403, 405, 410, 412, 414 and 418 veins) and the 500 vein zone (501 vein). The 501 vein has been barren to date, but 402, 403, 405, 410, 412, 414 and 418 have proven ore.

The significant point about these three south crosscuts is that 54 crosscut alone, which is essentially barren, would have failed completely to prospect the entire south area, and it is doubtful whether No. 3 crosscut would have aided in discovering the veins cut in 92 crosscut.

It is impossible to determine the number of veins or the amount of ore that might be credited to the use of this system of prospecting, but there is no doubt, after 3½ years of experience with it, that the structural conditions that prompted its use tend to hide ore bodies. This system has brought the geological conditions into focus so that they can be evaluated and the work can be planned for accomplishment in the most economical manner. The plan has already justified itself in new veins discovered and in savings effected in development by the proper arrangement of crosscutting.

## ACKNOWLEDGMENTS

The writer wishes to acknowledge the helpful criticism and suggestions of Mr. A. F. Duggleby, Mr. G. C. Worthington, Dr. R. M. Overbeck and Mr. B. C. Elsley. Thanks are given to Mr. R. B. Dumo for making the drawings.

# Applied Geology at the Magma Mine, Superior, Arizona

By Wilson D. Michell,* Member AIME

(New York Meeting, March 1947)

## Abstract

The Magma copper vein trends east-west, dips 70° south, and cuts through a 6000-ft thickness of limestones, quartzites, shale, diabase, and schist. The vein is itself a fault with a horizontal offset of 500 ft on the formation contacts in the two walls. There are two principal faults of large displacement and numerous smaller steep and flat faults that complicate the prediction of ore occurrence. A study of the geology, its systematic recording to facitliate ready reference, and graphic methods of interpretation of mineralization have aided the exploration, development, and mining of the ore. In making predictions of the distribution of mineable ore bodies and their shape in critical areas, studies are made of "mineralization contours" and "mineralization trends," which are developed on a longitudinal section of the vein from variations in the concentration of the combined base metal content as shown by assays of available openings in the mine.

## Introduction

This paper describes an example of geological work applied directly to mining and exploration problems in a deep vein deposit, and it is presented as being of interest especially to those engaged in working more or less similar veins.

In 1944 to 1945 the writer spent a year as assistant to J. K. Gustafson in a thorough geological examination of the Magma mine. Much of the writer's subsequent independent work described in the present paper had its inception in ideas originated by Gustafson during that examination.

## Objectives of Geological Work at Magma Mine

The objectives of geological work at Magma, as at operating vein mines of a similar nature, can be summarized as follows:

1. Direct assistance to the mining department by means of predictions as to the size, shape, grade, and location of ore bodies; by solution of fault problems and irregularities in the veins; by preparation of detailed ore-reserve estimates in specific areas; by keeping a complete record of the geology and assay results in the mine; and by guiding development work. In short, geology can aid in the realization of the prime purpose of the mining department, which is that of producing the greatest amount of ore in the most economical and expeditious manner possible.

2. Guidance for exploration, especially diamond drilling, in undeveloped areas of the mine.

3. Contributions to the understanding of the general geology of the deposit and its regional setting, including ore genesis, structure, and mineral relationships.

## Magma Ore Deposit

The east-west trending Magma vein, dipping about 70° south, cuts through a 6000-ft thickness of concordant limestones, quartzites, shale, diabase, and schist, named in descending order. These

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Manuscript received at the office of the Institute Jan. 17, 1947. Issued as TP 2214, Mining Technology, September 1947.

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rocks strike at right angles to the Magma vein and dip 30° east. The vein and its enclosing rocks are covered to the east by a great thickness of volcanics (dacite) of post-ore age.

The west portion of the Magma vein has been offset downward and southward by the west-dipping Main fault, the component of movement in each direction being nearly 1300 ft. About 2000 ft west of the Main fault the vein is terminated against dacite, which has been dropped on the west-dipping Concentrator fault, a fault of great magnitude. No exploration for the vein has been done west of the Concentrator fault.

The Magma vein is itself a fault, the south or hanging-wall block having moved downward relative to the footwall. This has resulted in a horizontal offset of about 500 ft on formation contacts in the two walls of the vein.

The principal ore body lies east of the Main fault and rakes steeply westward in the Magma vein. Some ore has also been mined in the segment of the vein between the Main and Concentrator faults, and smaller ore bodies of copper and zinc have been mined on the eastward extension of the vein under the dacite cover. The Magma vein has been explored more than 9000 ft along its strike, and the principal ore body has been mined 5800 ft downward on its pitch. Average thickness of the vein is 10 ft, with a maximum of about 30 ft. The Magma vein is not everywhere a simple unit structure, but in places splits into branches.

Ore-bearing portions of the Magma vein carry massive copper sulphides, principally bornite, chalcopyrite, and chalcocite, associated with pyrite in quartz gangue. Sphalerite is the ore mineral in the zinc areas of the mine.

The Koerner vein, similar in structure and mineral content to the Magma vein, but smaller, lies about 1100 ft south of the latter and is nearly parallel to it. The average thickness of the Koerner vein is less than 8 ft. The vein has no known surface outcrop.

Mining is by standard square-set cut-and-fill methods. The mine is very hot, so that work is made possible only by means of air conditioning, using underground refrigeration plants.

The most recent published complete description of the geology of the Magma mine is included in a bulletin of the Arizona Bureau of Mines by M. N. Short and others.[1] The geology of the Magma deposit has been worked out over a period of many years by numerous investigators, to whom Gustafson and the present writer are indebted for the basis of their own recent work. Complete reference to those previous studies is to be found in Short's bulletin.

## GEOLOGY APPLIED TO ORE PREDICTION

*Mineralization Contours.* It has been found desirable to predict for the mining department the distribution of mineable ore bodies on the vein in certain critical areas between levels (200-ft interval) and for lengths of 500 to 600 ft along the vein. In such places it was known that material of mining grade would not extend in an unbroken mass between levels, and that the outline of stoping ore in the plane of the vein would be an irregular line bordering tongues and embayments of low-grade material. In order to delineate such an ore outline in advance, assistance was obtained from a system of "mineralization contours" and "mineralization trends" on the Magma vein. The existence of this system was established by J. K. Gustafson, to whom credit is due for the application of the underlying principles to the Magma deposit.

The "mineralization contours" represent the relative concentrations of total base-metal content (copper, zinc, and lead) in the vein. At every point in the mine where assays were available covering full width of the Magma vein, the base-metal percentages were multiplied by the total width in feet, and the resulting products were plotted on a vertical longitudinal projection of the vein. An arbitrary scale of six subdivisions covering these values, ranging from 0 to 200 plus, was set up. The foot-assay values were then contoured on the basis of the established subdivisions, just as elevation points would be contoured at regular intervals in preparing a topographic map. A series of "ridges," or trends of high-metal content, separated by "troughs" of low-metal content became apparent. The ridges and troughs served to determine mineralization trend lines, and it is the trend lines that have been used in predicting ore shoots in the vein. The trend lines reveal a definite pattern of metal distribution in the Magma vein, and indicate the probable paths of ore solutions. The reasons for the localization of the high and low trends, and hence the explanation of the pattern, have not been discovered to date. Nevertheless, the existence of the trends is well established and can be put to practical use.

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[1] M. N. Short, et al.: Geology and Ore Deposits of the Superior Mining Area, Arizona. Ariz. Bur. Mines, *Bull.* 151 (1943).

Predicted ore distribution is based upon a study of all available assay results from development workings in the particular block in question, aided by the mineralization trend lines. In predicting the outline of mineable material on the vein in this way, it is always borne in mind that the mineralization trend lines and contours represent total metal content from wall to wall of the entire vein, and do not indicate grade of ore. Commercial ore, on the other hand, depends upon grade and thickness of individual strands within the vein zone. Therefore, the mineralization contours cannot be expected to give directly the shape and attitude of payable ore bodies; but since there is concordance in a general way between total metal content and grade of ore, the mineralization trends can be used to give guidance and direction in predicting the limits and position of ore shoots.

## PRACTICAL APPLICATIONS OF ORE PREDICTIONS

The most important and obvious use of predictions of the shape and attitude of ore shoots is as an aid to economical and efficient development planning. This is illustrated in Fig 1. At this locality, where the proposed work is now underway and where predictions appear to be reliable in a general way at least, the development procedure and planning would have had to be quite different if the margins of mineable ore had been otherwise located.

By means of detailed predictions of the shape and size of ore shoots, the reserve estimates can be refined to include individual stope blocks, thereby giving some idea in advance of the relative profitability of developing and mining a given area on the vein. Based on the predictions, reserve blocks are outlined with estimated tonnage and grade on large-scale vertical projections of the vein, together with the proposed preliminary development of the stoping area.

## METHODS OF RECORDING DATA

All data concerning ore predictions are recorded graphically, as far as possible, on plans, sections, and projections drawn on sheets of uniform size. Mineralization trends are transferred from small-scale vertical projections of the entire mine to 20-scale or 10-scale projections of the smaller area in question, and on these projections are shown assay results on margins of stopes and in development workings. Included also are important geological features, particularly faults. The predicted outline of mineable vein can then be drawn in projection on the

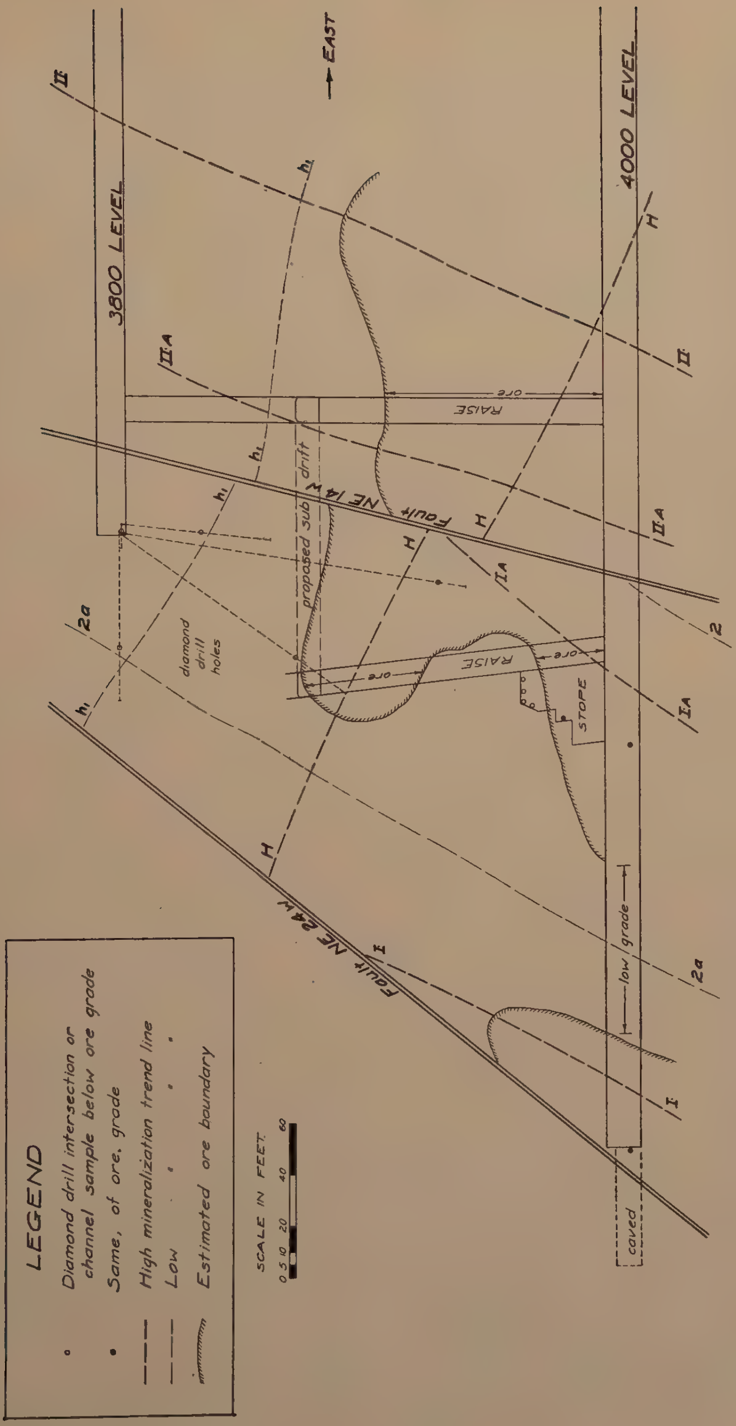

Fig 1—Mineralization trend lines and estimated ore boundaries in plane of Magma Vein. Longitudinal vertical projection.

same sheet with all the evidence on which the prediction is based.

Of importance in determining ore predictions and detailed reserve estimates is a routine record of current sampling in development raises. These assays, and also significant geological features, are plotted on 10-scale cross sections of the raises. Overall assay averages between levels are made from the cross sections and transferred to the longitudinal vertical projections.

### Geology as a Guide to Exploration

Exploration in the Magma mine is carried on principally by means of diamond drilling. An extensive exploratory drilling program was outlined by J. K. Gustafson in the report covering the geological examination previously referred to. Much of the proposed drilling was based upon extensions of the mineralization trends into unknown areas of the vein, thus providing specific favorable targets for drilling.

In certain areas of the mine, post-ore faulting of significant magnitude has taken place, It is important to determine the fault displacements as closely as possible, so as to project the position of the vein through the faulted areas and thereby direct the exploration. It is rarely possible, however, to obtain true displacements on the faults, but, by combining detailed north-south and east-west geological sections with level plans, a three-dimensional picture is obtained and the horizontal and vertical offsets on the vein are determined. In the case of the important Main fault it was possible to establish the true displacement, and thus estimate the location of the mineralization trend lines as well as other geological features in unexplored areas of the vein beyond the fault. We were provided by this means with definite drilling objectives.

Most of the logging of diamond-drill core at the Magma mine is done by the engineering department, the geologist being interested primarily in the geological interpretation and mapping of drilling results. All points of particular significance in the drill core, however, are checked by the geologist.

Drill-hole logs and assays of split core and sludge are recorded on special forms and filed in the engineering office. Diamond-drill core is reduced by selecting representative pieces, (except in vein, all of which is saved), and stored in readily accessible form in a core storage-house. This work is done by the engineering department. The geologist plots the drilling results on the geological plans, sections, and projections.

### Fault Studies as an Aid to Mining

In addition to the application of fault studies to exploration, the solution and mapping of faults have direct value as an aid to planning mine development and to the conduct of stoping operations.

*Important Steep Faults.* The Magma vein has been offset to varying degrees by steep faults cutting diagonally across the strike of the vein. Solution of a few of these having horizontal displacement on the vein of 50 ft or more is a problem largely concerning exploration, but other faults, especially those displacing the vein horizontally 15 or 20 ft, present obstacles to efficient mining operations. Faults which have displaced the vein sufficiently to cause fault gaps are particularly troublesome. Raises and stopes must be planned so as to permit extraction of ore against the fault on both sides without crossing through the fault itself, since such a procedure would result in much work in waste and bad ground.

It is apparent from the above that careful mapping and prediction by the geologist of the steep diagonal faults, and plotting them in plan, section, and longitudinal projection, are of assistance to the mining department.

*Flat Faults.* In certain areas of the mine a series of low-angle, or "flat," faults cut the Magma vein. The general char-

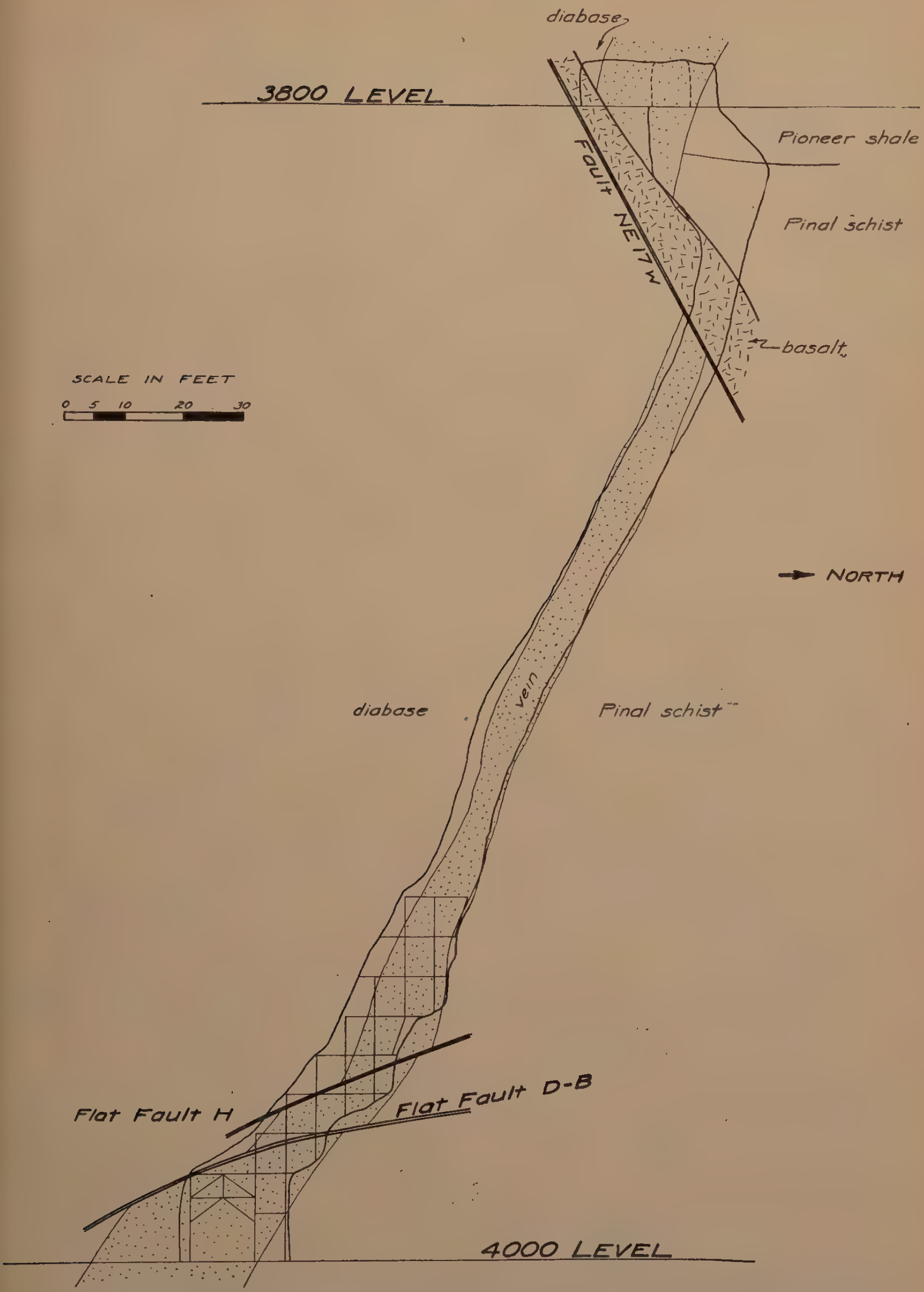

FIG 2—CROSS-SECTION THROUGH RAISE ON MAGMA VEIN, SHOWING OFFSET ON FLAT FAULTS.

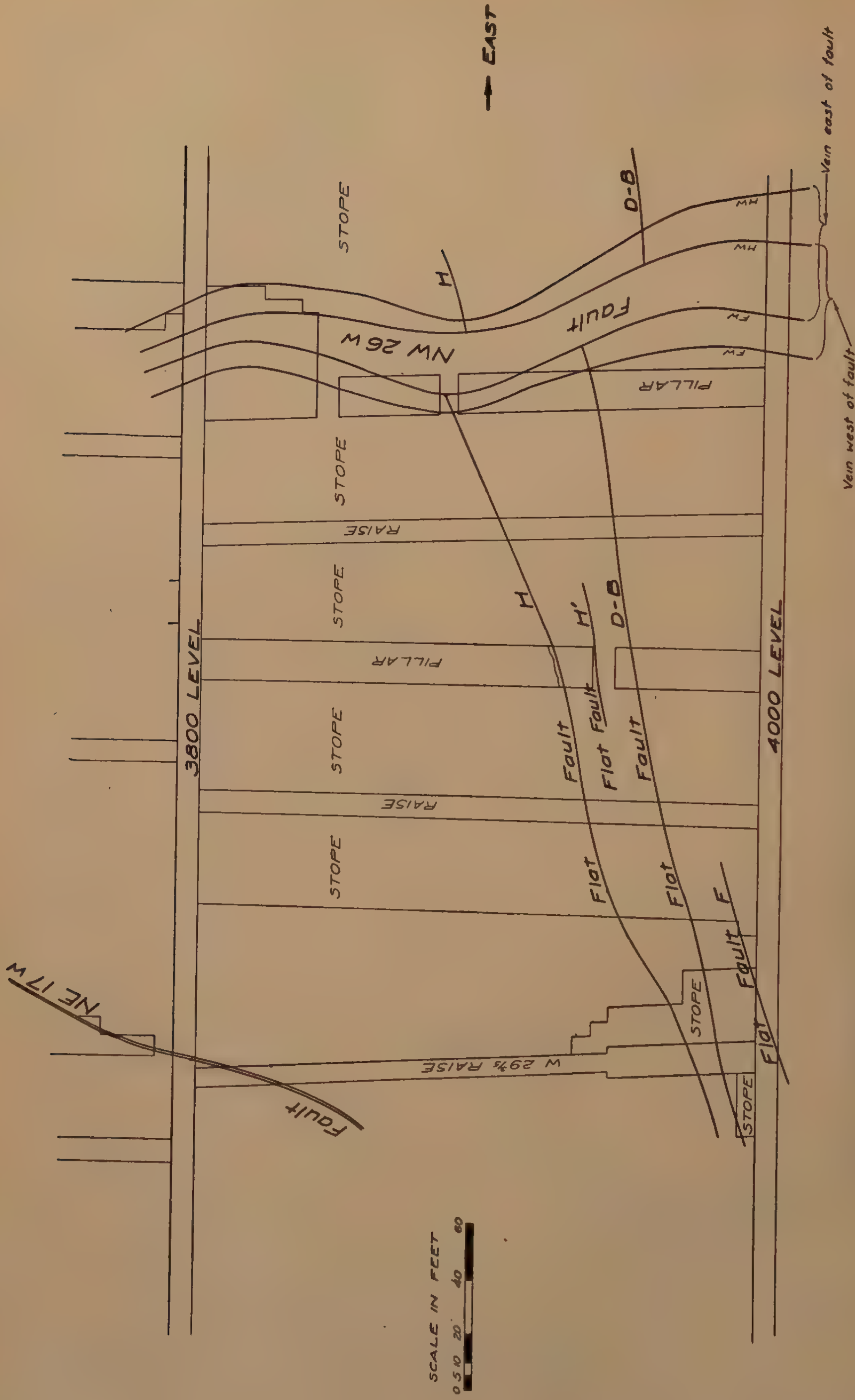

FIG 3—LONGITUDINAL VERTICAL PROJECTION OF PORTION OF MAGMA VEIN, SHOWING FLAT FAULTS.

acter of these faults was first described by Truman H. Kuhn in a geological report to the Magma Copper Co. in 1942, and summarized by him in Short's Arizona Bureau of Mines *Bulletin*.[1] The writer has continued the study of the flat faults as they have been encountered in current operations. Faults of this character are shown in Figs 2 and 3.

Various generalities apply to the flat faults: the average dip is 5° to 20° WSW, with minor rolls and reversals; the vein above the fault is in all cases offset northward (toward the footwall) 10 or 20 ft; the faults can be followed for several hundred feet east-west parallel to the vein, but because of their low angle of dip, they rarely cross the levels of the mine; some of the faults are mere flexures in the vein walls, whereas others are marked by slickensided and striated gouge, and the vein is cut off sharply; the striations and other evidence suggest the possibility of a considerable lateral component of movement; small subsidiary flat breaks are generally present beneath the larger faults, and serve to give warning as the main structures are approached by raises or stopes from below.

In some areas of the mine the vein proved to be consistently wider and of higher grade beneath the flat faults than it was above. This suggested to the writer that possibly some bending of the vein took place on the loci of the present flat faults before ore deposition, with the result that the places of flexure later served to localize the post-ore fault movement, which is now the most prominent feature. The pre-ore flexures might have slowed down or partially dammed ascending ore-bearing solutions, resulting in more concentrated deposition of metals beneath the obstacles.

The flat faults, although not major structures in the ore deposit, are important to mining because of the changes of width and grade associated with them; because they require awkward flattening of raises and stopes if they are not planned for in advance; and because they form zones of caving ground in raises and stopes, especially if they are encountered unexpectedly.

East-west longitudinal projections have proved most useful in plotting and projecting the low-angle faults (Fig 3). The longitudinal projections are based on information obtained in stopes and raises. Cross sections through the raises serve to indicate the amount of offset on the vein (Fig 2). By these means the faults have been predicted with fair accuracy in three dimensions throughout large areas of the mine.

*Methods of Analyzing Faults*. The absence of definite stratigraphic or other markers in large parts of the Magma mine has made it impossible to ascertain the true displacement on most of the faults, but the horizontal and vertical offsets of the vein can be determined, and, from a practical point of view, these elements are sufficient. By combining the geological level maps with cross sections and longitudinal sections at close intervals, and projecting and extending geological features boldly where necessary, the offsets on faults and a reasonably accurate three-dimensional picture can be obtained. Even if the interpolation and projection of geological features is great, care is taken that all three planes of section are in logical and accurate agreement.

In working out fault problems in the Magma deposit it is essential that individual faults be designated in such a way that they can be identified wherever encountered. This is done by the use of a system of nomenclature devised by J. K. Gustafson: a fault named thus, NE24w, is a northeast-trending fault, serial number 24, dipping westerly. By this method, individual faults can be identified, faults can be grouped for study according to general direction of strike and dip, and regardless of the apparent dip shown in a section oblique to the fault plane, the

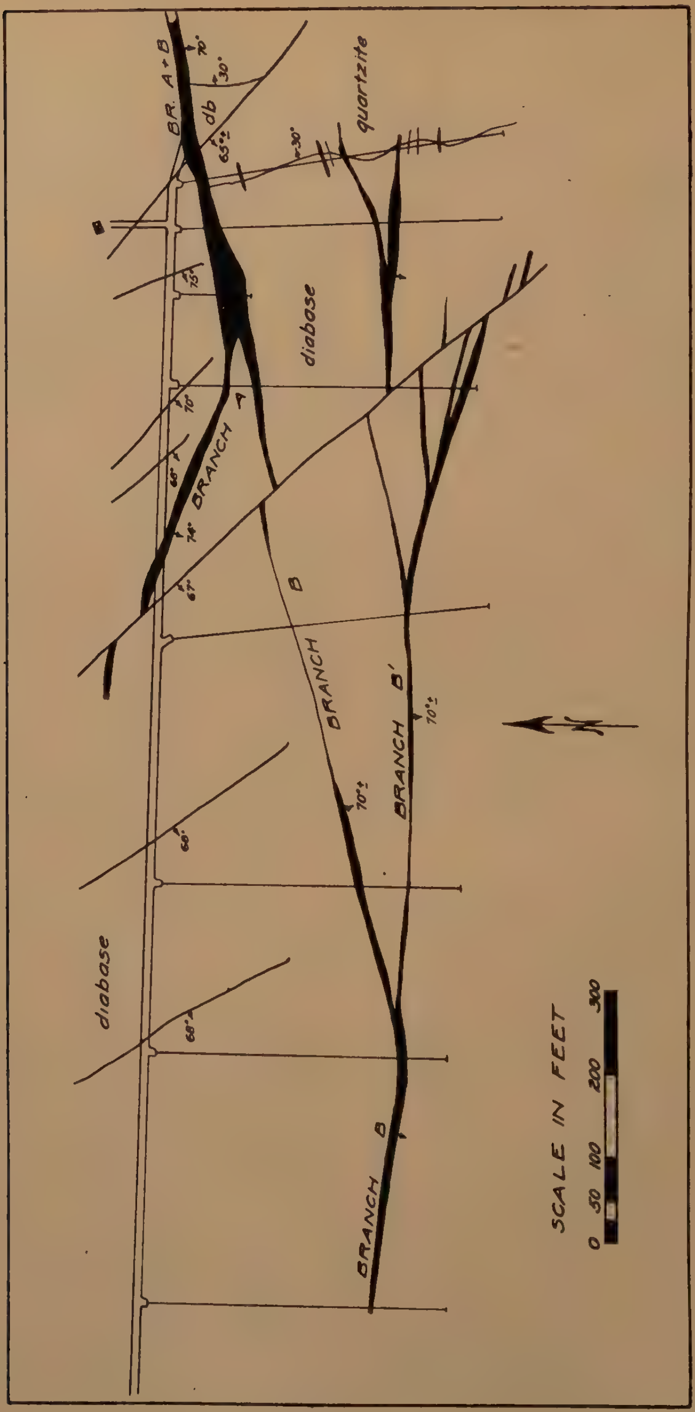

Fig 4—Branching in east portion of Magma Vein as shown by diamond drilling.

true attitude is clearly indicated by its designation. (See Figs 1 and 3.)

As has been stated, sufficient data were available to determine approximately the net displacement on the Main fault.* This was done by making, at 100-scale on the same system of coordinates, two north-south vertical projections (parallel to the strike of the fault), one being a projection of all geological features (formation contacts, faults, and vein) on the west or hanging wall of the Main fault, the other a projection on the east wall of the fault. The east-wall projection was made on transparent tracing paper. An equivalent point, in this instance the intersection of the Martin limestone-Troy quartzite contact with the south wall of the Magma vein, was identified on both walls of the Main fault. By moving the transparent sheet over the other one, in order to return the geological features to their original position before faulting, the vertical component of movement on the fault was measured as 1350 to 1550 ft and the horizontal component 1000 to 1300 ft, the hanging wall having been faulted downward and to the south, with a clockwise rotation (looking west) of about 5°.

Because of complications by minor faults and drag movement on both sides of the Main fault, a strictly precise figure for the net displacement could not be determined, but the method seems to be reliable and the results are within a useful range of accuracy. The procedure described can be employed in similar problems where the necessary original data are available.

Models provide a graphic illustration of fault problems in some instances. The writer made a model at a scale of 20 ft to the inch of a small critical portion of the mine between two levels. This was done in order to make the situation clear to mine bosses in charge of work in that area. Essential data on the two levels were drawn on sheets of glass supported in a light wooden frame. Faults were made of cardboard and the vein between levels was modeled in clay. When the usefulness of such a model has expired, the glass can be cleaned and the frame and glass can be utilized again for a model of some other part of the mine.

## Recording Geological Data

*Maps and Sections.* Observed geological features in the mine are mapped without interpolation or extension on 50-scale level plans, as has been done at the Magma mine for more than thirty years. These maps, which are on tracing cloth, are preserved in large looseleaf binders, and provide an invaluable basic record. They include observations made only on the levels, however.

For purposes of projecting geological features beyond actual mine workings and solving fault problems, the data from the 50-scale mine maps are transferred to 100-scale white prints of the surface and all levels. These are used as work sheets. The 100-scale maps were started for use in the preparation of Gustafson's general report. They are 5 by 3 ft in overall dimensions, and are filed in vertical hanging racks in the vault. Longitudinal sections and cross sections at close intervals on coordinate lines, also at 100-scale, have been constructed on white prints of the same size as the level plans. It is from these working drawings, in three mutually perpendicular planes, that major structural problems are worked out.

In addition, combined geological and assay details, prepared primarily for the mining department, are made at 10-scale and 20-scale. These maps are of standard size, 33 in. horizontal by 27 in. vertical, overall. The size was chosen to permit easy handling, to allow for filing in vertical hanging racks of 3-ft width in the mine superintendent's office and in the engineering office, to cover the 200-ft level interval even at 10-scale, and to fit the 36-in. white print paper most commonly used in the printing machine. The 10-scale and 20-scale detailed drawings consist of geological and assay data plotted on plans and composite plans of the levels, on vertical cross sections through all development raises and on coordinate lines at critical locations, and on vertical longitudinal projections of the vein showing stoped areas, mineralization trends, predicted ore outlines, detailed ore reserves, and others. These large-scale drawings are prepared at the request of the mine superintendent to cover areas of the mine in which particular problems are being encountered in current operations.

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* This study was made with J. K. Gustafson in connection with his geological report on the Magma mine.

A separate card catalogue has been prepared for the detailed maps just described. Copies of this index, on 5 by 3 filing cards, are kept with the maps in both the engineering office and the mine superintendent's office. The cards are filed by location in the mine, so that all drawings that have been made of a particular place can be readily located. A sample card, to show the amount of information which is kept in the files, is reproduced below:

*Loc.* 3200 Level — *Map No.* 10-GA-9
E 16⅗ Rse.
*Eng. File No.* HR-C-4
Geologic and Assay Section of
East 3200 16⅗ Raise.
*Scale:* 1″–10′ — *Date:* Feb. 8, 1946
*Size:* Standard — *Revised:* 4/10/46
*Made by:* W. D. M. — *Filed:* Eng., Mine

The map number, 10-GA-9, is interpreted as follows: 10-scale, geological-assay drawing, serial number 9. The maps are hung in groups according to the scale and type indicated by the map numbers, and in serial order within each group. After looking up a map in the location index, it can be immediately selected from its group in the hanging racks by means of the map serial number.

The engineering file number, HR-C-4, is interpreted thus: Hanging rack, section C, hanger number 4. Each hanger contains a separate group of 10-scale or 20-scale maps, in serial order within the group. This reference locates the original of the desired drawing in the engineering office vault, and also makes it possible to key in the geological-assay map index with the index of all mine maps on file in the engineering office. Prints of the geological-assay maps are filed and catalogued in exactly the same way in the mine superintendent's office.

*Diamond-drilling Records.* Detailed logs of diamond-drill holes, including sludge assays, split-core assays, and accepted assay results are kept on special forms. The core is reduced for storage by selecting representative pieces of all rock types, although complete split core is saved in the vein. The core is filed in a special storage house in boxes that are easily handled and readily accessible.

The holes and the features encountered in them are plotted on the various geological maps and sections and are used in the solution of geological problems and the prediction of ore (Figs 1 and 4).

*Reports and Specimen Collection.* The writer has prepared semi-annual reports for interested company officials on the progress of geological work. These reports cover all aspects of the work, with brief discussion of problems under consideration and a summary of results. Special illustrations are prepared to clarify the material.

A reference collection is kept of representative or unusual rock and ore types from the Magma mine and vicinity. The specimens are placed with labels in envelopes or small boxes, and are numbered and stored in serial order. There is a check list with brief description of each specimen. At present the writer is preparing a card index in which the specimen descriptions are filed by rock types, formations, or chief ore minerals represented. A sample index card is shown below:

*Dripping Spring quartzite* *No.* 16

*Loc.:* North of Silver King Wash (see surface geological map)

*Descr.:* Slabby, red, arkosic quartzite.

*Coll. by:* J. K. G., W. D. M. *Date:* 4/20/44

## ACKNOWLEDGMENTS

Thanks are due to J. K. Gustafson, of Newmont Exploration Limited, for criticism of the manuscript and for permission to describe various ideas originating with him; to W. P. Goss, General Manager, and J. F. Buchanan, Mine Superintendent, Magma Copper Co., for helpful discussion of problems involved.

# Structural Control of Copper Mineralization, Bagdad, Arizona

By Charles A. Anderson,* Member AIME

## Abstract

The Bagdad copper deposit is of the disseminated type (porphyry copper) occurring in a quartz monzonite stock of late Cretaceous or early Tertiary age. This stock, located essentially at the intersection of two dike swarms, is the only one of several stocks cropping out in the area which is appreciably fractured and mineralized. Most of the rocks in the area are of pre-Cambrian age, but their structures have had no discernible influence on the copper mineralization. Hypogene alteration of the quartz monzonite includes the addition of quartz, orthoclase, and sericite and recrystallization of biotite. The hypogene sulphides are pyrite, chalcopyrite, and molybdenite, the latter younger than the chalcopyrite. The late Cretaceous or early Tertiary igneous rocks, veins, and faults were controlled by northeast and northwest shear zones, the intersection of which marks the zone of appreciable copper mineralization. The minor mineralized fractures carrying the bulk of the copper have a preferred orientation parallel to the intersecting shears, and it is concluded that they are of tectonic origin rather than the result of shrinkage on cooling, crystallization, or mineralization. Supergene enrichment during late Tertiary or early Pleistocene time formed a chalcocite zone providing the minable copper ore. The enrichment was controlled in large part by northeast and northwest faults, particularly at their intersection or where the faults are closely spaced.

## Introduction

A major objective in a regional geologic study of a mining district is to determine whether any structural pattern can be found that has a bearing on the control of mineralization. The Bagdad copper deposit is of the disseminated type (porphyry copper) occurring in a quartz monzonite stock which is located essentially at the intersection of two dike swarms (Fig 2). Of several quartz monzonite stocks in the area only this one is appreciably fractured and mineralized. The origin of the minute mineralized fractures in disseminated copper deposits has long been a challenging problem to the mining geologist, and evidence has been obtained to suggest that the mineralized fractures at Bagdad are related to a regional pattern of conjugate shears which also controlled the location of the dike swarms. The purpose of this paper is to present this evidence and to emphasize the structural control of the supergene enrichment in forming a chalcocite zone that has provided the ore for the Bagdad mine.

The field work on which these conclusions are based was done between October 1943 and August 1945, as a part of the copper investigations of the Goelogical Survey, U. S. Department of the Interior. The writer and his associates, E. A. Scholz and J. D. Strobell, Jr., have prepared a comprehensive report on the geology and ore deposits of the Bagdad area which will be published subsequently by the Geological Survey.

Bagdad is located in west-central Arizona in Yavapai County. Ransome[1] divided the State of Arizona into three provinces: I, the plateau to the northeast; III, the desert region to the southwest and II, the moun-

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Published by permission of the Director, U. S. Geological Survey. Presented at the Arizona Section meeting, Tuscon, Oct. 28–30 1946. Manuscript received at the office of the Institute April 19, 1947. Issued as TP 2352 in Mining Technology, March 1948.

* Geologist, U. S. Geological Survey, Prescott, Ariz.

[1] References are at the end of the paper.

tain region which lies between them (Fig 1). Bagdad lies in the northwest part of the mountain region, approximately in the middle of the belt.

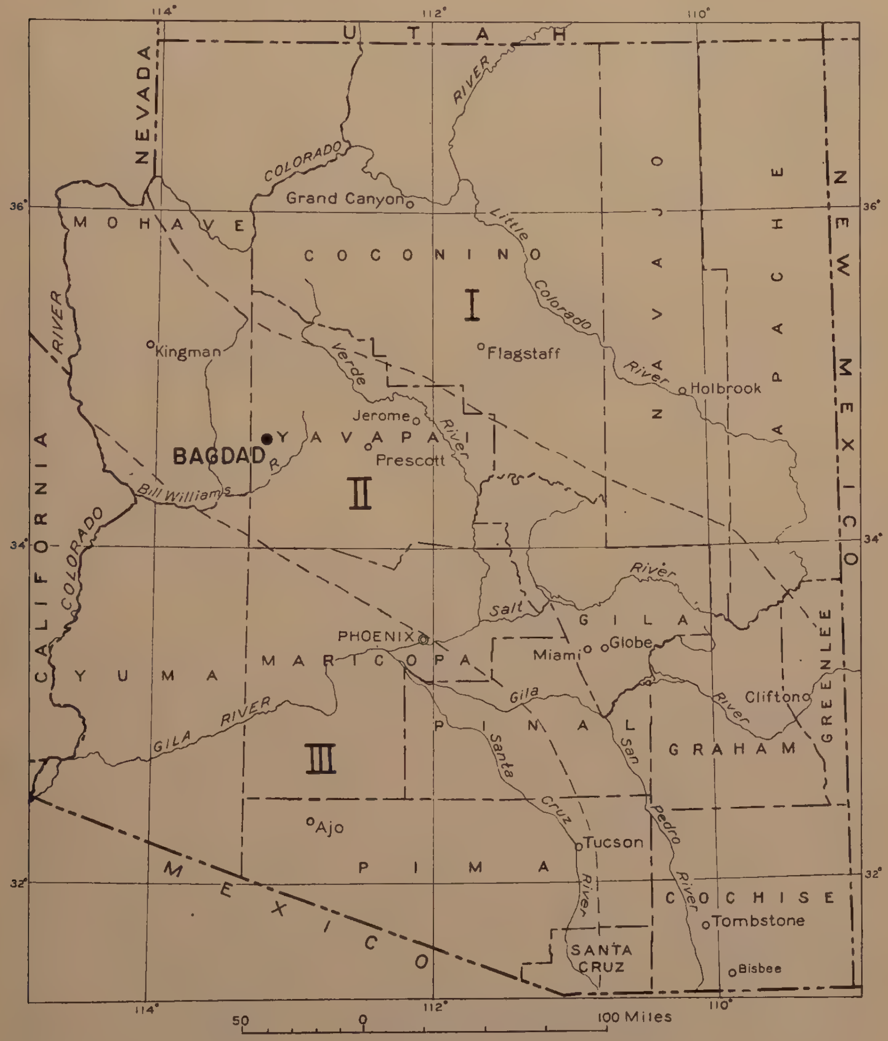

Fig 1—Index map of Arizona.

## General Geology

Most of the rocks exposed in the Bagdad area are of pre-Cambrian age (Fig 2) and consist of amphibolites, various schists, and slates, intruded by a diverse series of pre-Cambrian igneous rocks ranging from meta-rhyolite, meta-gabbro to granite. The structure of the amphibolites and schists is complicated by local close folding and some pre-Cambrian faulting. Pre-Cambrian granite crops out extensively in the area and is believed to occur beneath most of the amphibolites and schists so that structurally the pre-Cambrian rocks are essentially massive and except for some

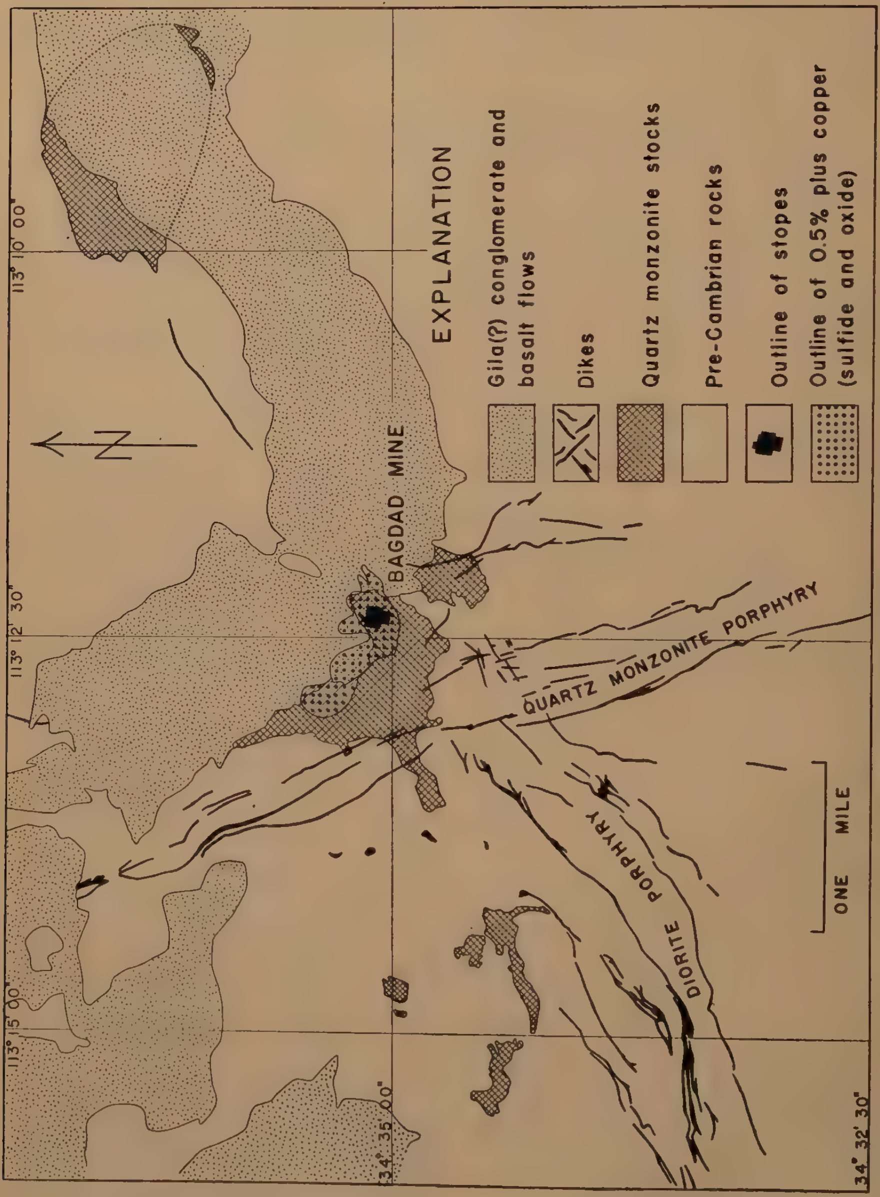

FIG 2—GENERALIZED GEOLOGIC MAP OF THE BAGDAD AREA.

local control of the dikes appearing in the schists, the pre-Cambrian rocks and structures have had no apparent control on the late Cretaceous or early Tertiary igneous intrusions and disseminated copper mineralization.

A small erosional remnant of massive rhyolite tuff remains in the southwestern corner of the Bagdad area, resting on the pre-Cambrian rocks (not indicated on Fig 2). By analogy with similar rocks elsewhere in Arizona, this tuff is probably late Cretaceous or early Tertiary in age (Laramide revolution). The rhyolite tuff is intruded by rhyolite dikes that are in turn intruded by quartz monzonite, suggesting that the latter rock and associated dikes and mineralization are also of late Cretaceous or early Tertiary age. The quartz monzonite crops out in a series of stocks and plugs. Dikes of diorite porphyry and of quartz monzonite porphyry are younger than the quartz monzonite and form the intersecting dike swarms.

During late Tertiary or early Pleistocene a surface of considerable relief was partly buried by the Gila(?) conglomerate and its intercalated basalt flows (Fig 2). The lava flows now cap extensive mesas.

## Hypogene Alteration and Mineralization

Much of the quartz monzonite in the mineralized area has a rather fresh appearance and contains little or no conspicuous chlorite and sericite to suggest the usual type of hypogene alteration. Petrographic studies of the quartz monzonite from the mineralized area and adjacent unmineralized areas disclosed that the hypogene alteration involved albitization of the plagioclase feldspar, introduction of granular orthoclase and quartz, and the recrystallization of biotite to leafy aggregates of lighter-colored biotite. Microscopic sericite is present in some of the albitized plagioclase and appears also in veinlets associated with quartz and orthoclase. Only small local areas are intensely silicified and sericitized. Apatite and rutile were formed during the period of alteration.

Pyrite and chalcopyrite occur in disconnected granules, averaging 1 mm or less in dimension. These are concentrated largely along veinlets spaced a fraction of an inch to three inches apart and associated with introduced quartz, orthoclase, and sericite. Some of the sulphide granules are truly disseminated but they are always associated with quartz and orthoclase. Quartz-pyrite-chalcopyrite veins are present, one-eighth to one inch wide, locally widening to three inches. Two veins (Fig 3), however, have a width as great as 20 in. The veins are composed of glassy to white, vuggy quartz in which pyrite and chalcopyrite appear as scattered aggregates or as vug linings. Sphalerite and galena are mineralogical curiosities appearing as small aggregates in some of the wider quartz-pyrite-chalcopyrite veins. Quartz-orthoclase-pyrite-molybdenite veins cut the pyrite-chalcopyrite veins indicating that the molybdenite is younger than the chalcopyrite.

An examination of many polished surfaces of the primary sulphides collected from various parts of the mine gave a general impression that the total content of chalcopyrite plus pyrite in the minor fractures or disseminated through the rock is about the same irrespective of the grade of copper as shown by corresponding drift assays. The proportion of chalcopyrite to pyrite varies, however, and, in the zones of lower copper grade, pyrite is in excess of chalcopyrite whereas the reverse ratio holds true where the copper grade is higher.

## Late Cretaceous or Early Tertiary Structure

The quartz monzonite stocks crop out in a belt 7 miles long with an approximate trend of N 70° E (Fig 2). The limitation of these stocks to this belt is indicative of a fundamental structure control, and the intrusion of the stocks marks the first major episode in the late Cretaceous or early Tertiary structural history. Several small aplite dikes in the central stock at Bagdad have a N 40° W trend. The diorite porphyry dikes which cut the quartz monzonite locally, have a generalized trend of N 60° E though some variation is present particularly to the southwest where these dikes in part follow the structures in the pre-Cambrian schists. But where they extend northeast into the massive granite, the dikes are approximately parallel to the general trend of the quartz monzonite stocks. Locally a north trend is present where the dikes follow older faults. Although no direct evidence is available, by analogy with the usual sequence of rock types in and around centers of igneous intrusion, the inference is drawn that the aplite dikes are older than the diorite porphyry dikes. The diorite porphyry dikes are cut by the quartz monzonite porphyry dikes which have a generalized trend of N 20° W, but with the scale of the field mapping used and nature of the exposures, no measurable displacements were proved. The quartz monzonite porphyry dikes are cut by pyrite-chalcopyrite-galena veins in the southern part of the area, indicating that the dike swarms were intruded prior to mineralization in the Bagdad area.

At the time the Bagdad mine was mapped, December 1943 to January 1944, four levels were accessible, 30 ft apart vertically. In the upper three levels, boundary drifts were accessible along 10 stope boundaries driven essentially on the 100-ft coordinate lines as part of the stope preparation for block caving. On the lower or haulage level, drifts connecting the draw-points made the lower part of the ore body almost completely accessible. A total of 836 fractures, showing gouge or crushed rock indicating some movement, were mapped as well as recognizable quartz veins, 367 in number, having sufficient continuity to determine attitude. The minute fractures along which the bulk of the sulphides were deposited could not be mapped underground as only two dimensions could be recognized on most of the drift walls and their lack of continuity made it impossible to determine the third dimension. On the surface over the mine, the attitude of many of these minor mineralized fractures could be determined owing to the irregularities of the outcrop surface and oxidation of the sulphides which clearly showed the trace of the fracture on the outcrop. The attitude of 472 mineralized fractures was determined from random points over the mine.

A convenient method for comparing many strikes and dips of veins and faults is to plot the attitudes on equal area projection nets, for in this method the three dimensions can be plotted on a two dimensional diagram. An excellent description of this method is given by Billings.[2] Plotting of the veins on an equal area projection discloses three maxima. The greatest number of veins strike N 60° E and dip steeply northwest and southeast. The second maximum has a strike of N 10° E and steep westerly dips. The third maximum is at N 40° W with steep southwest and northeast dips. Crosscutting relations indicate that the N 60° E veins are younger than the N 10° E veins, but no positive evidence could be found relating to the age of the N 40° W veins. Most of the quartz-molybdenite veins have random orientation and flatter dips, as low as 25°, and they are younger than the northeast veins. The two wide quartz-pyrite-chalcopyrite veins shown on Fig 3 strike N 40° W and dip 65° to 70° SW and the adjacent aplite dike has the same strike but steeper dip, 80° to 85° SW.

The minor mineralized fractures are not at random as a casual examination would suggest, but three maxima stand out on the equal area projection. The greatest number are concentrated at N 70° E and dip 70° N. The second maximum is at N 20° E and dips 75° NW and the third maximum is at N 40° W and dips vertically. In general a close correlation in attitude exists between the dominant trends of the minor mineralized fractures and the larger veins, except that a northerly dip is recorded for the dominant N 70° E fractures and steep dips in opposite directions are recorded for the N 60° E veins.

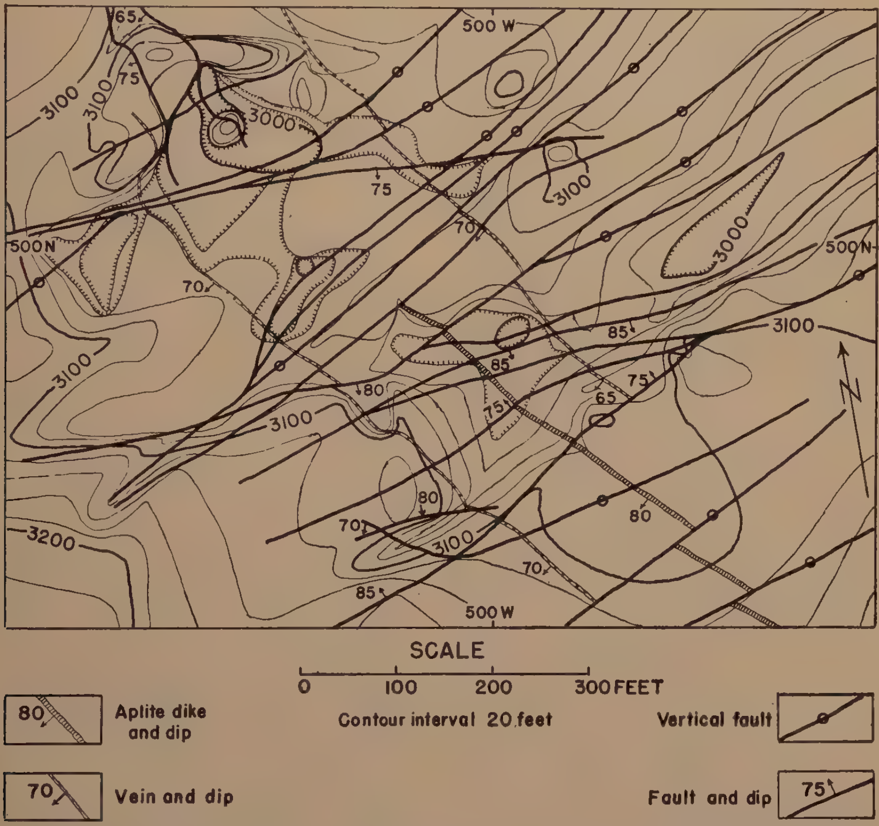

FIG 3—CONTOUR MAP OF THE BASE OF THE CHALCOCITE ZONE SHOWING LARGER FAULTS, VEINS AND APLITE DIKE.
Where no information is available intersections are shown without offset.

Faults striking northeast are of importance because of the number having this trend and because they show the greatest continuity and greatest amount of displacement (Fig 3). The dominant strike is N 70° E with dips ranging from northwest to southeast. Commonly these faults followed the veins, in places completely brecciating the vein or in other places following the hanging wall or footwall of the vein. These northeast faults displace the aplite dike, the stronger northwest veins, and most of the northwest faults. The greatest offset observed was 35 ft where one of the faults cut the aplite dike, but of the 62 intersections of northeast faults with older faults or veins, the displacement was one foot or less in 83 pct of the observations. A few had displacements as large as 5 to 10 ft. The sum total displacement along all of the northeast faults is low; assuming that the strike of the aplite dike is constant, the total eastward displacement is 100 ft, distributed along 12 of the larger faults. An additional 20 to 30 ft is suggested by the displaced northwest veins, indicating that the total eastward displacement through a distance of 1000 ft was less than 150 ft. Some of the displacements conceivably could be explained by dip-slip movement offsetting high-angle faults or veins but in most intersections a strike-slip component was obvious as faults or veins of opposite dip were offset the same amount and in the same direction. Numerous slickensides have a pitch of 10° to 20° E coinciding with the observed offsets indicating that although the net slip was oblique, the strike-slip component was much greater than the dip-slip component. In general the movement on the northeast faults was to the northeast and down on the north side, regardless of the direction of the dip of the fault.

Some faults have a northwest strike, but they are subordinate in number to the northeast faults and most of them have no great continuity except in the northwest corner of the ore body (Fig 3). Usually the northeast faults cut the northwest faults but in a few examples the reverse was noted. The northwest veins were re-opened by faults that appear to be older than the northeast faults. With one exception where a northeast vein was offset 5 ft, most of the displacements along northwest faults are one foot or less. Where the slip components can be determined the strike-slip component was northwest on the northeast side. In general the strike-slip component was greater than the dip-slip component.

TABLE 1—*Trends of Structures*

| N 60°–70° E | N 40° W | N 20° W | N 10°–20° E |
|---|---|---|---|
| Stocks | Aplite dikes | Quartz monzonite porphyry dikes | Veins<br>Minor fractures |
| Quartz diorite porphyry dikes | Veins | | |
| Veins | Minor fractures | | |
| Minor fractures | Faults | | |
| Faults | | | |

The period of faulting was largely pre-Gila(?) and pre-supergene enrichment, but some post-Gila(?) movement was noted locally where Gila(?) conglomerate was dropped several feet on the north side of one of the northeast faults.

The prevailing northeast and northwest trends of the faults, veins, and minor fractures are essentially parallel to the structural trends related to the quartz monzonite stocks and associated dikes (Table 1).

The angular difference of 20° between the N 20° W dikes and N 40° W structures, is not great and presumably these two trends represent essentially parallel structures. However the N 10° to 20° E trend of one set of veins and mineralized fractures is not represented by any of the older or younger structures.

Only the northeast faults yield positive evidence of the character of rupture, and as stated above, the movement has been east and slightly down on the north side of the faults indicating that faulting took place as a result of a nearly horizontal shear couple and the north side moved east relative to the south side. It might be inferred that the earlier parallel fractures occupied by northeast veins, diorite porphyry dikes and quartz monzonite stocks formed also from a horizontal shear couple.

The suggestion might be offered that the northwest fractures, particularly those occupied by the aplite dikes and the wider northwest veins were formed by tensional stress but no direct evidence was found to support this interpretation. The fact that some of the northwest faults have an oblique-slip movement, up and to the northwest on the northeast side is more suggestive of another essentially horizontal shear direction. Furthermore the general alternation of rupturing in one quadrant, then the other, does not favor the suggestion of tension in the northwest quadrant, but rather that it was a zone of secondary shear, the two quadrants of rupture representing conjugate shears in which the strike-slip component is greater than the dip-slip component.

No data are available to offer suggestions concerning the N 10° to 20° E trend.

## CONTROL OF THE HYPOGENE MINERALIZATION

Hulin[3] has suggested that shrinkage following cooling and crystallization is responsible for the minor fracturing in the porphyry copper deposits, and Locke[4] has emphasized the possibility that shrinkage during mineralization is an important factor. For the Ajo deposit Gilluly[5] considered these possibilities, adding collapse owing to escape of magma or volatiles from a lower horizon or to volcanic explosions that overcame the pressure of the cover rocks, concluding however that the ultimate cause of brecciation in the Ajo deposit could not be determined from the present information. Pennebaker[6] in discussing the Ely porphyry copper deposit, made the pertinent comment that if shrinkage on cooling is responsible for the fracturing, all of the porphyry bodies at Ely should be similarly fractured, whereas only the porphyry that is within the ore zone is broken. He concluded that tectonic activity was the fundamental cause of minor fracturing in the Ely deposit. Butler and Wilson[7] pointed out that northeast fractures at Morenci determined the path along which the porphyry stocks were intruded and that continuation of these stresses produced breaks in the solidified porphyry, now occupied by veins. They suggest that the minor fractures might fall into a definite system if mapped in detail.

If shrinkage following cooling, and crystallization, were of paramount importance in producing the minor fractures at Bagdad, the other stocks in the same area should be expected to show the same kind of fracturing and since they are unfractured and unmineralized, serious doubt exists as to the validity of this explanation for the Bagdad deposit. The fact that the pattern of these minor fractures is essentially parallel to the pattern of the other structural features such as the dikes, veins, and faults, is more suggestive of tectonic stresses as the ultimate cause of fracturing. However, the more random and less numerous quartz-molybdenite veins, in places having low dips, occupy fractures that may owe their origin to some other cause such as subsidence following mineralization.

From a broader view, the intersection of the two shear zones, graphically illustrated by the intersection of the dike swarms, is probably of greatest importance in marking the favorable zone of copper mineralization. If this interpretation is correct no prospecting targets for new ore bodies of the porphyry copper type are indicated in the Bagdad area. This conclusion is supported by a lack of any visible signs of appreciable copper mineralization in the other exposed stocks.

## SUPERGENE MINERALIZATION

A chalcocite zone has been formed by supergene enrichment in the hypogene mineralized quartz monzonite. This enrichment took place prior to or during the deposition of the Gila(?) conglomerate, and the top of the chalcocite zone slopes northward in general parallel to the base of the Gila(?). The chalcocite zone has been truncated by the present erosion surface (Fig 4). Exploratory churn-drill holes give a general picture of the distribution, grade, and thickness of the chalcocite zone, and the best known portion of the zone is being mined now by the Bagdad Copper Corp. The chalcocite zone was formed by downward movement of copper-bearing solutions which have replaced chalcopyrite and pyrite by chalcocite, leaving a leached zone above of variable thickness, locally reaching 350 ft.

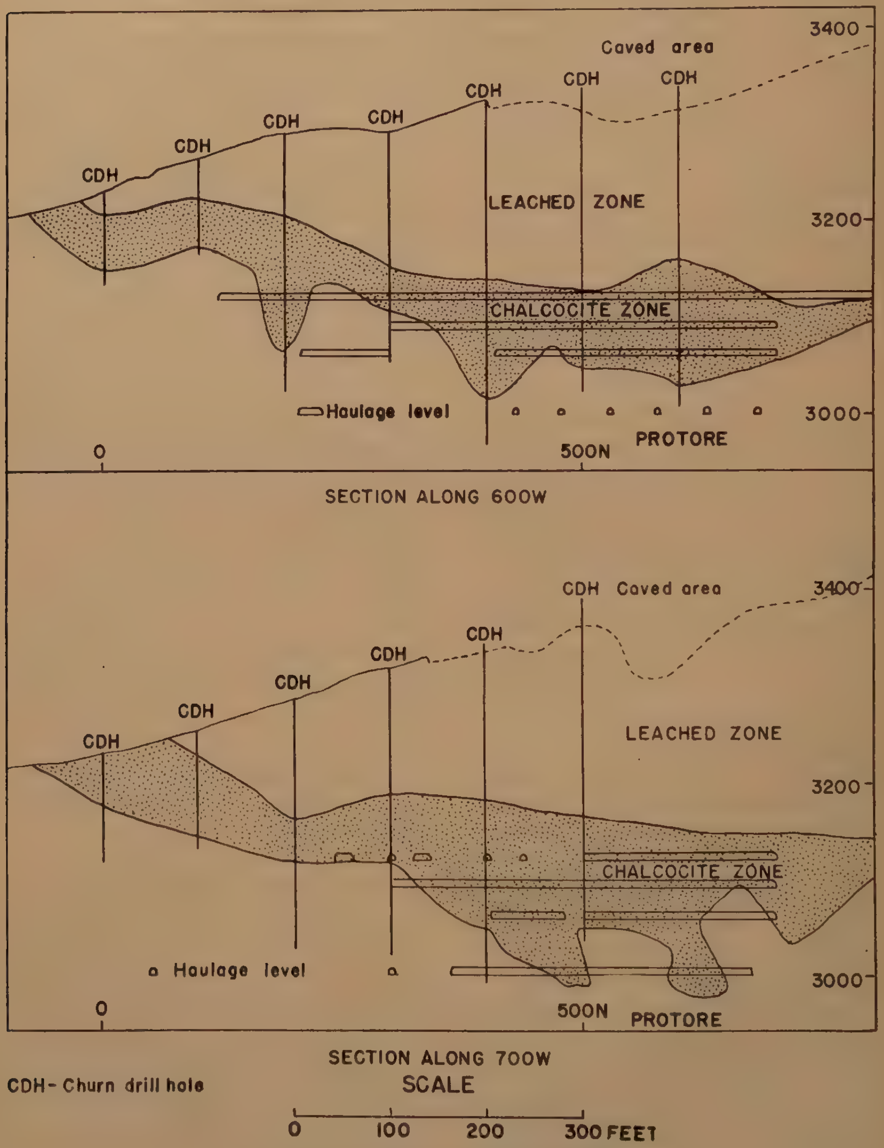

FIG 4—SECTIONS THROUGH THE BAGDAD MINE.

Some local concentration of oxidized copper minerals, chiefly malachite and chrysocolla, has taken place in the leached zone, but in general the copper content in this zone is less than in the protore.

Typical chalcocite ore consists of grains of gray chalcocite 1 mm or less in dimension, present along fractures or disseminated in quartz monzonite. The quartz monzonite in the supergene zone has a duller luster than in the primary sulphide zone. Thin sections reveal little change in the chalcocitized quartz monzonite except that some of the feldspars are partly clouded with minute indeterminate spots of clay(?). Biotite and sericite are unaltered.

Microscopic studies of the ore minerals from the chalcocite zone reveal that chalcocite replaces chalcopyrite in preference to pyrite. In good chalcocite ore, the pyrite may be replaced partly, or show no signs of replacement. Indeed, no polished surfaces of chalcocite ore failed to reveal some unreplaced pyrite. However, in the wider quartz-pyrite-chalcopyrite veins where post-mineral faulting increased permeability, both sulphides are completely replaced by chalcocite near the top of the zone, and sooty chalcocite is rather common.

The geometry of the chalcocite zone in the Bagdad mine is fairly well known because of the close spacing of churn-drill holes and accessible boundary drifts at the time of our underground mapping. The visual study of the ore was aided greatly by systematic drift assays. These data were compiled on numerous cross-sections, two of which are reproduced (Fig 4). The top of the chalcocite zone can be connected only from drill hole to drill hole, so it is drawn more regularly than it appears in detail in the limited exposures underground or in the glory hole. The position of the base of the chalcocite zone could be determined with more precision because of the numerous underground openings, and as illustrated in Fig 4, it is more irregular than the top. A contour map of the base is shown in Fig 3. The thickness of the chalcocite zone is variable, averaging about 100 ft but ranging from 20 to 180 ft, the thickest parts usually appearing where the elevations of the base are lowest (Fig 4).

Actually the base of the chalcocite zone is not as sharp as cross-sections imply, for a transition zone exists from good chalcocite ore to the primary sulphide zone (protore) in which no chalcocite can be recognized. In this transition zone, usually 10 to 20-ft thick, chalcocite coatings appear on the chalcopyrite. The selection of the base of the chalcocite zone is thus necessarily arbitrary and in this study it was placed near the top of the transition zone where chalcopyrite was conspicuous and chalcocite is present only as thin coatings. In actual practice this zone is marked by a decrease in grade of copper from above 1 pct to around 0.8 pct. In the protore where no visible chalcocite is present, the grade drops to 0.5 to 0.7 pct copper. In other words, the "chalcocite zone" as used in this study applies to the zone in which appreciable enrichment in copper has occurred.

### CONTROL OF THE SUPERGENE MINERALIZATION

Observations underground disclosed that good chalcocite ore appeared on the hanging wall of many minor faults and protore was present on the footwall. The reverse relationship was noted in only a few places. This fact would suggest two conclusions; (1) post-chalcocite faulting, or (2) the permeability of the quartz monzonite was increased by the faults and this was the important factor in determining the depth to which chalcocite enrichment took place. Without denying the possibility of some post-chalcocite faulting, and locally such faulting took place, the measurable dip-slip components along these faults are not of the right order of magnitude to explain the difference in thickness of the chalcocite ore. The larger faults in which continuity could be established have been plotted on the contour map of the base of the chalcocite zone (Fig 3) and the relationship between these faults and the lower elevations of the base is very striking. In the northwest corner of the ore body, northwest faults intersect northeast faults and a thick section of high-grade chalcocite ore was found. In the central part of the mine (Fig. 3) the base of the chalcocite zone rises appreciably to the southwest where the several strands of the faults join, but to the northeast where the faults spray out in horsetail fashion, troughs of chalcocite ore appear. The importance of faults and fractures in controlling supergene enrichment at Miami and Ray has been noted by Ransome.[1]

## DISTRIBUTION OF COPPER

In order to obtain a picture of the distribution of copper in the mineralized stock, the grade of total copper, both oxide and sulphide, to 500 ft below the bedrock surface was determined from the churn-drill and diamond-drill holes. For some of the shallow holes, estimation of the grade was necessary for the lower part of the 500-ft section and assays from adjacent deeper drill holes or underground workings were used for this interpolation. The outline of the block containing 0.5 pct or more of copper has been plotted on Fig. 2. The outline of the stopes of the Bagdad mine is also shown, this position marking the place of highest grade copper. Although the details of the boundary as plotted may be changed by new data, scout churn-drill holes block out the essential outline in sufficient detail for the scale of Fig 2.

The northwest and northeast trends are clearly shown in the outline of the better grade copper and it is significant that the zone of highest grade of copper, indicated by the Bagdad stopes, is present essentially at the intersection of the northwest and northeast trend of the outline (Fig 2) and parallel to the northeast and northwest intersecting shear zones.

## CONCLUSION

Two intersecting shear zones in which the strike-slip component is greater than the dip-slip component controlled the intrusion of a series of quartz monzonite stocks, related dikes, and subsequent veins and faults. The minor mineralized fractures carrying the bulk of the copper were formed by tectonic stress and the central stock at Bagdad, located at the intersection of the shear zones, is the only stock that is appreciably fractured and mineralized with copper. Supergene enrichment was controlled by faults parallel to the two shear zones and the thickest and highest-grade chalcocite ore occurs where intersecting or closely spaced faults have increased permeability.

## ACKNOWLEDGMENTS

Thanks are due Mr. E. R. Dickie, General Manager of the Bagdad Copper Corporation, for permission to publish data obtained from the company records. To my associates in the Bagdad project, Mr. E. A. Scholz and Mr. J. D. Strobell, Jr., grateful acknowledgment is made for valuable field assistance and stimulating debate. The manuscript has been reviewed by Dr. James Gilluly, of the University of California, Los Angeles; Dr. Hugh E. McKinstry, of Harvard University; and Mr. S. C. Creasey, of the U. S. Geological Survey, and I am indebted to them for helpful comments.

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# San Manuel Prospect

By H. J. Steele* and G. R. Rubly,† Members AIME

## Location and History

The San Manuel property is in townships 8S and 9S, range 16E, Gila and Salt River Base and Meridian, State of Arizona. This area is in the Old Hat mining district, southern Pinal County, and is about 3½ miles southwest of the town of Mammoth on the San Pedro River and one mile south of the Mammoth mine at Tiger, Arizona. The property consists of over 150 contiguous lode claims.

The Old Hat district was prospected prior to the Civil War, but there was little or no production until 1881.[1] The chief producers have been the Mammoth and Mohawk mines, now consolidated and operating as the St. Anthony Mining and Development Co., Ltd. Production to 1936 was $5,200,000, of which 83 pct was gold.[2] Since 1940 the production has been chiefly lead and zinc, and some molybdenum and vanadium. There are a few small properties in the vicinity that have produced very minor tonnages of copper ore.

In the San Manuel group there are claims which have been held continuously from 1906 to the present time, and at least two churn-drill holes were put down on or near the outcrop in 1917. No logs for these holes are available, but it is assumed that the grade of copper encountered was not sufficient to encourage further exploration.

A small amount of ore has been shipped from open cuts on the San Manuel outcrop area, but there is no record of the tonnage or grade. It is estimated that not more than 50 tons were mined. These cuts, plus a few shallow shafts and some short adits indicate limited and unenthusiastic prospecting at San Manuel.

In 1942, through the efforts of James M. Douglas, R. B. Giffin, Victor Erickson, and Henry W. Nichols, owners of the San Manuel group, the Reconstruction Finance Corporation and War Production Board authorized the United States Geological Survey to investigate the property, and by their recommendation, the United States Bureau of Mines was authorized to put down a limited number of churn-drill holes. The Bureau of Mines carried on drilling from November 1943 to February 1945. Seventeen holes were drilled, varying from 305 to 1990 ft in depth.

The property was brought to the attention of Magma Copper Co. in June 1944, and an option agreement was made with the owners. Subsequently, a group of claims lying north of the original group and held by the Apex Lead Vanadium Mining Corp., and a group to the east held by the Quarelli family, were optioned. A drilling program was laid out to conform to the Bureau of Mines pattern, and drilling started in December 1944. San Manuel Copper Corp. was incorporated September 1945, and has carried on the exploration program to the present time.

## Geology

### *General*

The mineralized outcrop, chief copper mineral of which is chrysocolla, is a tri-

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Presented at the Arizona Section Meeting, Tucson, Oct. 28–30, 1946. Manuscript received at the office of the Institute Nov. 12, 1946. Issued as TP 2255 in Mining Technology, September 1947.

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[1] References are at the end of the paper.

angular area with a northeast-southwest base of 380 ft and with the apex of the triangle 400 ft to the southeast. Northwest of the base is Red Hill, so called because of the prominent gossan coloration. With a few exceptions, conglomerate and alluvium cover the rest of the property. The elevation varies from 2900 to 3500 ft above sea level. The area slopes gently to the east, with the exception of a small rugged and precipitous portion around the south side of Red Hill, and is cut by numerous dry washes with easterly courses toward the San Pedro River. The cover is sparse brush and cacti characteristic of this portion of the state.

### *Rocks*

Relatively few rock types are found in surface exposures or are encountered in drilling. These formations were mapped by Dr. G. M. Schwartz of the United States Geological Survey, and the logging of all drill holes has conformed to his classification.[3] The rocks are quartz monzonite, monzonite porphyry, diabase, felsite, and felsite breccia, Gila conglomerate and recent alluvium. An aplitic phase of quartz monzonite is abundant, particularly to the southwest, toward Oracle.

The quartz monzonite found in the drilling area is granitic in texture, showing varying degrees of hydrothermal alteration. Quartz is abundant; the feldspars and biotite are extremely altered in most cases. Rutile is common, and the rock in general is sericitized and kaolinized.

The monzonite porphyry has a fine groundmass composed largely of a mosaic of quartz and feldspars. There is an abundance of biotite but less quartz than in the quartz monzonite. The feldspar phenocrysts are usually well altered, and the biotite is often bleached or chloritized. Secondary quartz flooding is fairly common, especially near the ore zone. A distinct phase of dark gray, fine-grained porphyry, containing an abundance of biotite in the form of phenocrysts recrystallized to aggregates of grains and much disseminated secondary hydrobiotite in the groundmass, is found in the hanging wall of the ore body to the south of the outcrop area.

Alteration products found in thin section in both the quartz monzonite and monzonite porphyry, as reported by G. M. Schwartz, include the following minerals: the hydromicas, allophane, epidote, leucoxene, montomorillonite, dickite, chlorite, potash clays, and calcite.

Diabase occurs as dikes cutting both the quartz monzonite and monzonite porphyry. The rock is well altered, grading from fine to coarse diabasic texture, and is mineralized with chalcopyrite and pyrite, or with chrysocolla and iron oxides in the oxidized zone.

The felsite, which is unmineralized and which cuts the diabase and the monzonites, is light gray to light cream in color, fine-grained, with occasional small quartz and feldspar phenocrysts in a finely crystalline groundmass.

The diabase occurs infrequently, and the felsite, while more abundant than diabase, is relatively unimportant in the over-all rock distribution.

The Gila conglomerate consists of alluvial material interbedded with lava flows, breccias, and tuffs. The coarse conglomerate on San Manuel is composed of boulders of acidic and basic volcanics cemented with clay. A phase of the Gila conglomerate, designated as granitic conglomerate and composed largely of fragments of relatively unaltered quartz monzonite, has been found in drilling to the southwest.

### *Structure*

The structure at San Manuel, interpreted largely through drilling results, can only be generalized at the present time.

The Gila conglomerate, which covers the major part of San Manuel, strikes northwest and dips northeast at 20° to

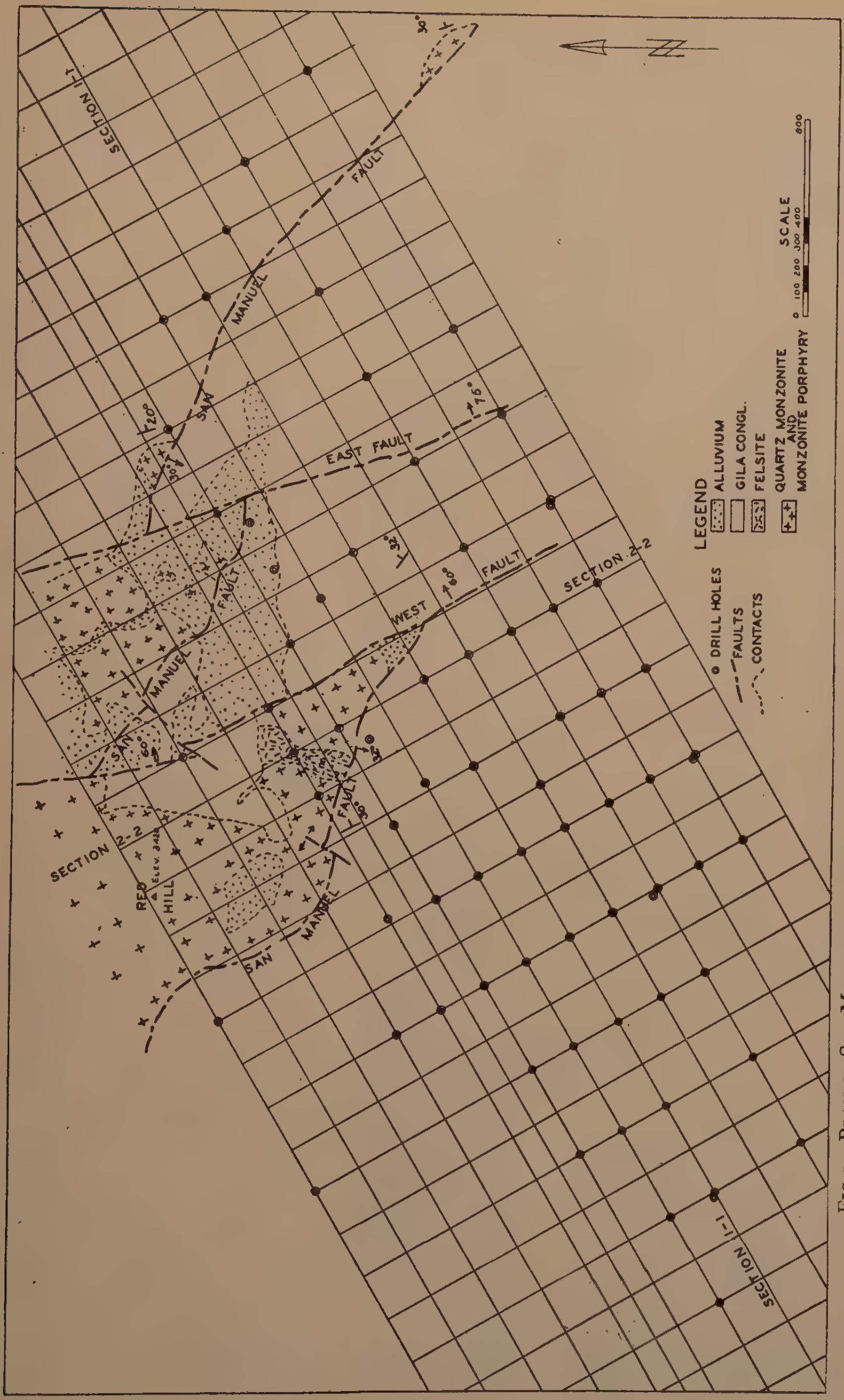

FIG 1—PLAN OF SAN MANUEL PROSPECT SHOWING DRILLING GRID, CHURN-DRILL HOLES AND AREAL GEOLOGY.

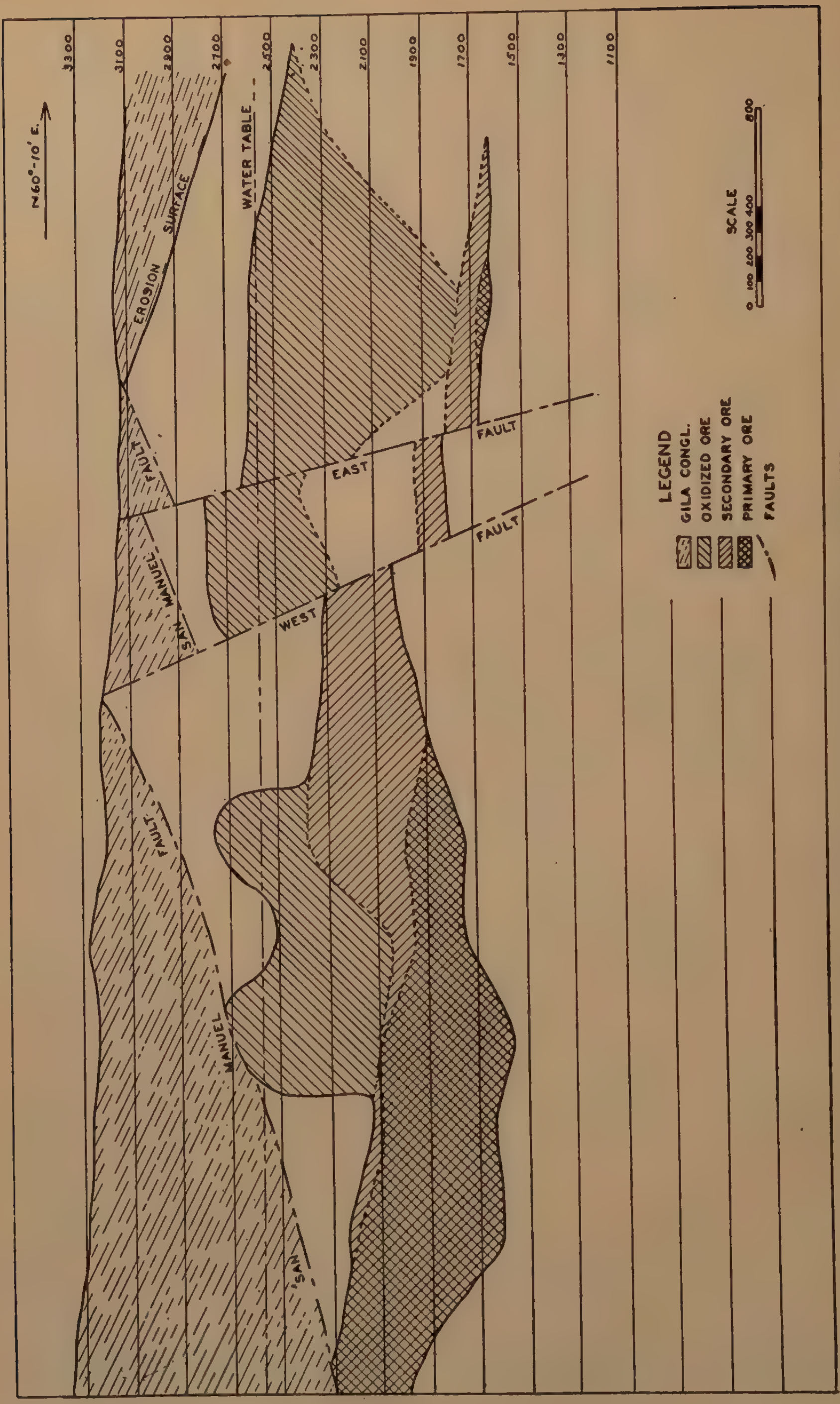

FIG 2—SAN MANUEL PROSPECT, LONGITUDINAL SECTION, I–I.

45°. The San Manuel fault, striking northwest, more or less parallel to the strike of the conglomerate, and dipping southwest at 25° to 35°, forms the contact of the monzonite and the conglomerate. The San Manuel fault is in turn cut by two 60° to 75° northeast-dipping faults striking northwest. The most westerly of these faults is designated as the west fault, and the most easterly as the east fault (Fig 1). This step faulting of the conglomerate is illustrated in Fig 2, and has been substantiated by drilling to the northeast. Minor outcrops of hematite-stained, altered monzonite porphyry occur at intervals along the surface exposure of the most northeasterly segment of the San Manuel fault. Drilling northeast of these exposures indicates that the contact of the conglomerate with the old erosion surface dips northeasterly at 20° to 25°. The indicated vertical displacement of the west fault is in the neighborhood of 400 ft, and of the east fault, 200 ft. The east fault has been considered to be the southward extension of the Mammoth fault as mapped by Peterson[4] and others. A more complex fault system may be expected and is indicated by the drilling, but correlation of evidence of faulting is not possible from the churn-drill results.

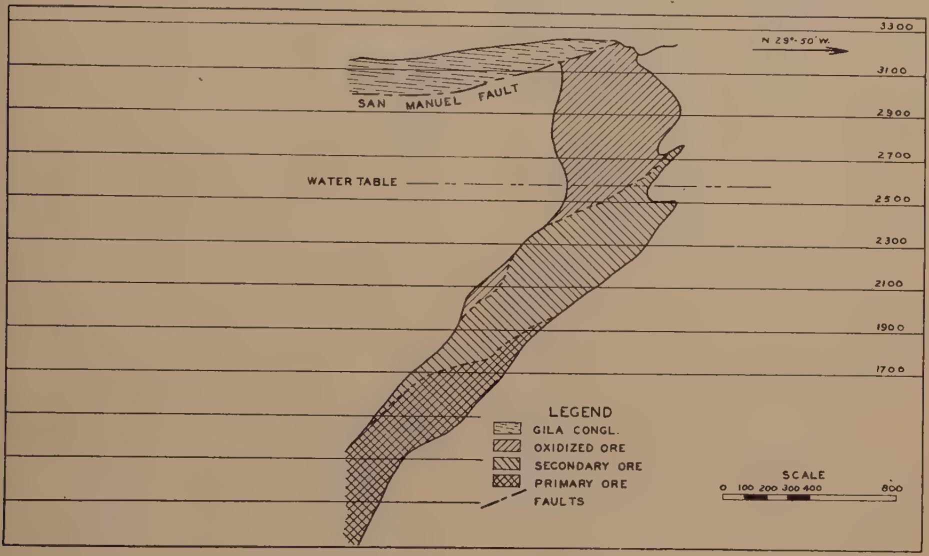

FIG 3—SAN MANUEL PROSPECT, CROSS SECTION, 2–2.

The outcrop area, where copper mineralization is found (Fig 1), is bounded on the southwest leg by the San Manuel fault and on the northeast leg by the west fault. The base, as indicated in cross section in Fig 3, is controlled by decreasing copper mineralization, and is therefore, an assay boundary. A diabase dike, striking northeast and nearly coinciding with this cutoff of copper mineralization, was originally thought to be a diabase filled pre-Gila fault which abruptly cut off the northwestern extension of the ore body. Drilling northwest of this line has indicated a gradational decrease of copper mineralization in the projected ore zone rather than faulting.

The persistency of the San Manuel fault on both strike and dip is unusual, and it is difficult to visualize a low angle normal fault of this magnitude. The most feasable explanation, with the limited data at hand, can best be presented by this possible sequence of events:

1. Intrusion of the monzonite, and mineralization of the intrusive masses.
2. Oxidation and secondary enrichment.
3. Deposition of the Gila conglomerate. (Events 2 and 3 may be contemporaneous).
4. San Manuel fault formed, dipping 60° to 65° to the southwest.
5. Regional tilt of 25° to 35° to the northeast and development of the fault system, which strikes northwest, and dips to the east.

The San Manuel ore body has been described as a tabular mass, striking north 60° east, and dipping southeast at 50° to 60°; and is shown this way in Figs 2 and 3. It must be remembered, however, that this strike and dip is largely caused by an assay boundary. Any arbitrary cutoff figure selected would change the attitude of the ore body. The strong pyritic mineralization below the outlined ore body has not been bottomed by any drilling to date. The upper portion of the hanging-wall rock is very lightly mineralized; alteration and mineralization increases at depth until the zone of better copper mineralization is encountered.

Copper mineralization has been proved by drilling for 5000 ft in the northeast-southwest direction, and 1400 ft in the northwest-southeast direction. Pyritic mineralization is known for 2200 ft northwest of the ore zone.

### *Mineralogy*

In the oxidized zone the chief ore mineral is chrysocolla; cuprite is present locally; malachite and azurite are not abundant. The chrysocolla is generally found in joints and fracture planes.

Chalcocite is the predominate copper mineral in the secondary zone. Residual primary chalcopyrite, occurring chiefly as intergrowths with the secondary minerals, and secondary chalcopyrite are found. Some native copper is found as well as minor amounts of bornite and covellite. There is very conclusive evidence of one or more stages of partial oxidation of the secondary zone in parts of the ore body. This may be caused by variations of the water table and regional tilting.

Chalcopyrite and minor amounts of secondary chalcocite are the chief copper minerals of the primary zone. The proportion of chalcopyrite to pyrite determines the footwall of the ore body. Slight oxidation is found relatively deep in the primary zone, and is thought to coincide with fault planes. The sulphide minerals are well disseminated throughout the monzonite, as well as occurring as veinlets. Chalcopyrite and chalcocite are frequently found as films on pyrite. Molybdenite is present in sufficient amount to be of economic importance, and can be seen megascopically in a large proportion of samples in the primary zone. Gold and silver are found in small quantities throughout the deposit, but sufficient work has not been done toward determination of the gold and silver content to state the average assay at this time. Rutile is fairly abundant throughout, and is thought to be present as an alteration product of biotite.

Polished sections of the concentrates, selected at intervals in the sulphide zone, were prepared and examined by G. M. Schwartz of the United States Geological Survey.

## CHURN DRILLING

### *General*

A grid coordinate system, bearing 29° — 50′ west of true north, controls the churn-drilling pattern. The orientation was set by the Bureau of Mines, and was based on the strike of the apparent northwesterly limit of copper mineralization of the outcrop. The drilling done by the Bureau of Mines and the early drilling by San Manuel Copper Corp. was spaced rather widely over the area to establish probable continuity of the ore body and to determine the required spacing of holes for future drilling. When intermediate holes were drilled and the general attitude of the ore zone became apparent, the present drilling pattern evolved. This pattern is based on a 400-ft interval in the direction of strike, which is from east to west on the drilling grid, and a 200-ft interval in the north-south direction. All holes since the beginning of exploration have been controlled by the grid co-ordinate system. The few exceptions are the result of rugged terrain making the correct location impractical.

The drilling grid is extended from permanent reference points by transit and tape traverse; elevations are carried to the collar of each hole by a line of levels from established reference points. Elevation datum is from the United States Coast and Geodetic Survey bench marks. The drilling grid is tied to the San Manuel co-ordinate system, which is based on a true north meridian. Holes are spudded in as near the intersection of grid co-ordinate lines as practical, and the error in hole location is rarely more than one foot in either direction.

### *Statistics*

At the present time 67 holes have been completed, and 11 are in progress for a total of 110,854 ft of drilling since exploration of the property began. The Bureau of Mines drilled 15,824 ft, and San Manuel Copper Corp. has drilled 95,030 ft. The deepest hole drilled to date has been 2600 ft. All drilling is done on contract, with the contractor furnishing all equipment and tools.

The average drilling conditions at San Manuel are illustrated by the following tabulation of statistics for a three-month period from Sept. 1, 1945 to Nov. 30, 1945. Drilling shifts refer to time actually spent in drilling; total shifts worked refer to drilling shifts plus the time spent moving and setting up on new locations, repairs, fishing, and miscellaneous lost time:

TABLE 1—*Average Drilling Conditions at San Manuel*

| Drilling Conditions | Average | Best Drill | Poorest Drill |
|---|---|---|---|
| Elapsed time (days) | 91 | 91 | 91 |
| Number of drills operating | 6 | 1 | 1 |
| Total footage drilled | 13,296 | 3,439 | 71 |
| Drilling shifts | 698.6 | 116.5 | 13.8 |
| Advance per drilling shift (ft) | 19.0 | 29.5 | 5.1 |
| Total shifts worked | 825 | 133.8 | 23.7 |
| Advance per shift worked (ft) | 16.2 | 25.7 | 3.0 |

The average time required to move to a new location was 4.6 shifts. This includes pulling casing from the completed hole, moving, and rigging up preparatory to spudding in.

### *Equipment*

The selection of churn drills for exploring this property was based on the record of accurate tonnage and grade estimates computed from data obtained by churn drilling at other well-known disseminated copper deposits. Exploration by diamond drill was rejected because of probable low core recovery if the formations proved to be highly fractured.

Several types of churn drills are operating on the property; Bucyrus-Erie machines being in predominance. Models 24-W, 24-L, 29-W, 29-T, and 36-L, with drilling capacities ranging from 1500 ft to 3600 ft are represented. These are all-steel machines, very compactly built, and ideal for working in rough terrain. The space required for a setup is comparatively small, and the machines can be handled over fairly steep grades. There are four Ft. Worth Spudders in operation: Models Super D, F, Jumbo H, and Super J; having drilling capacities ranging from 2500 to 5000 ft. These are larger machines of wood construction, and require considerably more room for a setup, as the engine is usually set back of the rig some 20 to 30 ft. Power transmission is by flat belt drive. They are more awkward to move, and take longer to set up and tear down than the Bucyrus-Erie machines.

The drilling contractors, operating under similar contracts, hire all help required to run the rigs. A crew usually consists of a driller, a tool dresser, and welder. Since all bits are built up by electric welding, the tool dresser is more of a general helper than his name implies. Some contractors work on a three 8-hr shift basis; others on two 12-hr shifts. All drills are now operating 24 hr per day, and 7 days a week.

### *Drilling*

As the early drilling was comparatively shallow, holes 8-in. in diameter were started, reduction to 6 and 4-in. holes then being possible. When it became evident that deeper drilling would be required, with the possibility of having to case and reduce size of hole several times before reaching bottom, the starting hole was increased to 10-in. and finally to 12-in. diameter. This permits running 10-in. casing followed by 8-in., 6-in., and 4-in. when necessary.

There has been considerable discussion regarding the advisability of starting holes with 12-in. bits. It was thought that the larger diameter hole and the greater shock produced by heavier tools would increase caving in fractured ground. The drill contractors preferred the 8-in. bit because a larger bit would probably slow the drilling speed in the upper portion of the hole, where contract prices per foot are relatively low. It has been our experience, however, that the larger sized hole does not decrease speed of drilling greatly, and that caving would be encountered at about the same place in the hole regardless of size of bit used. The larger hole gives one better assurance of being able to drill to the depth required. The 12-in. hole is never carried to more than 1200 ft, and casing is usually required before this depth is reached. The 10-in. hole is carried to 1500 or 1600 ft if possible, allowing the hole to be completed with either an 8-in. or 6-in. bit. The deepest hole, which was 2600 ft, was finished with a 4-in. bit after having set a 400 ft liner of 4 in. pipe at 2400 ft. Some of the ground seems to stand well, as three holes, the deepest of which was 2200 ft, have been drilled without the use of any casing.

Drill locations and access roads are constructed to meet the requirements of the type of machine that will occupy the site. The Bucyrus-Erie machines dump the sludge on the left side, facing the machine, while the Ft. Worths dump on the right side; thus necessitating a different type of location. All roads and locations have been made by bulldozer and scarifier, on contract; no drilling and blasting have been required.

Fishing for lost tools is taken care of by the contractors, with the exception of some long difficult jobs below 1000 ft, in which case the company has assumed part of the burden of cost. This situation has not come up often.

Unless casing is ordered by the field engineer to shut off possible caving and the resultant contamination of samples, especially in or near the ore zone, its use is usually left to the discretion of the contractors. The contractor is required to make every effort to recover all of the casing, but there are times when several joints have to be cut off the bottom in order to salvage any. This is accomplished by setting off a bomb, fired electrically at a coupling near the bottom, or by using a "collar buster"; a device lowered on the string of tools to a predetermined depth, which punctures and slits the pipe until a collar is reached and cut. The bombs are not too satisfactory because they tend to swell the casing in one or more places, making withdrawal of the pipe difficult.

The use of any drilling muds or Aquajell is, in general, prohibited. The exceptions to this rule are to permit the use of Aquajell, plastered on casing couplings when being run into the hole to make later withdrawal easier, and during fishing operations in caving ground. In crooked holes it is occasionally necessary to drop surface rocks in the hole and redrill in order to straighten it. In any of these operations the driller is required to thoroughly clean the hole before sampling operations are resumed.

In order to accurately establish the depth from which samples are coming, the holes are measured frequently with a special measuring device consisting of a reel of 3⁄32-in. galvanized aircord marked with lead buttons every 10 ft. The 100-ft marks are identified by a code of elongated buttons. The measuring cable was calibrated by attaching a 7-lb lead weight to the zero end; lowering it down a vertical shaft; and marking off the units on surface. A very definite jerk is felt when the weight is pulled off the bottom of the hole.

Frequency of measurement varies. Holes are measured when they near the contact of the conglomerate and monzonite, so this point can be closely determined. After that, measurements are made at about 200-ft intervals, and again when the hole is completed.

## SAMPLING

### *Equipment*

Sampling equipment consists of a dump box, a 6-ft launder, a sample splitter, and 5-gal milk cans to catch the samples.

The dump boxes are built of 2 by 12 in. plank, lined with 1⁄4 in. steel plate welded at the joints. The boxes are 12 in. wide, 36 in. long, and 24 in. high (inside dimensions). The launder is constructed of 1⁄8 in. steel plate, and designed to fit securely, with a 3-in. overlap between the steel lining and wood exterior, at the open end of the dump box. The discharge end of the launder lies on the upper edge of the splitter unit. The slope of the launder is adjusted to insure an even flow of sludge to the splitter, which is a modification of the Ray Consolidated Co. splitter.[5] It is, essentially, three or more Jones splitters arranged in series to facilitate gravity flow of the sludge through the successive individual units. The upper two tiers of slots of a four-cut splitter are 1¼ in. wide, and the lower two tiers are ¾ in. wide; the slots being staggered by ½. The sample portion of the sludge passes straight down and is discharged at the front. The rejects gather underneath and are discharged at the side.

### *Personnel*

It is desirable to have as samplers young engineers or geologists. This type of help has not been available, so it has been necessary to employ less skilled men, which in turn has meant special training and supervision for the sampling crew. Every detail of catching, preparing, washing character samples, and panning for concentrates had to be taught in most cases. Sampling is supervised by shift bosses and engineers who visit the rigs periodically during sampling operations.

### *Technique*

The churn-drill holes are sampled at 5-ft intervals, from the contact of the conglomerate and the monzonite, to the bottom of the hole. Character samples are taken at 20-ft intervals in the conglomerate.

The sludge is dumped from the bailer into the dump box, and flows through the splitter. The driller is cautioned to ease the bailer into the box to avoid surge and splash. The sample portion is caught in a 5-gal milk can and a bucket of rejects is caught; a portion of which is washed free of slimes and saved as a character sample. A uniform volume of rejects from each

5-ft interval is panned to obtain a concentrate which is dried and sacked. An engineer examines both the character sample and concentrate to determine, megascopically, the rock type, alteration, kind, and relative amounts of mineralization. The character samples are filed for future reference, and the concentrate is sent to G. M. Schwartz of the United States Geological Survey for selected samples to be polished and examined by microscopic methods.

The sampler fills out a "churn-drill report," noting the depths at which the samples are caught, color of sludge, condition of hole, depth at which water was encountered, fault zones as evidenced by gouge, and any other pertinent data relative to the hole. The spaces for rock type and mineralization are filled in by the engineer who examines the character and concentrate samples.

The size to which the sample is cut depends largely on the size of hole being drilled, and if drilling is under water. A 12½-in. hole may require a cut to 1/256 of the total volume of sludge bailed, and a 4-in. hole is cut to ⅛ or ¼ of the original volume of sludge. The sample is cut so there is a minimum of about 15 lb of dry solids.

Drillers are required to bail holes clean. This is relatively simple above the water table, and takes about 3 or 4 trips into the hole. However, after the holes are under water, it is never possible to get a really clean hole, as there is always a certain amount of slimes in suspension. In this case, a few tests are made to determine how many bailings are required to pick up the largest part of the sample, and leave the hole relatively clean for the next run. On numerous occasions it has been possible to obtain samples of the slimes in suspension by carefully bailing a hole after it had been shut down for periods of 24 to 48 hr. Assays of this material have agreed very closely with the assay of the last run made, so that no serious errors are introduced by leaving this material in the hole. No attempt is made to shut the water off by casing.

### Drying and Preparing Pulps

The samples caught in milk cans are gathered up, usually at the end of the shift, and hauled to a centrally located drying room. They are dried on a series of nine heavy steel drying tables, each about 3 ft square. Steam, which is generated to 20 lb gauge pressure in a manually-operated vertical fire tube boiler, using diesel oil for fuel, supplies the heat for the drying tables.

Other drying room equipment consists of a Jones dry splitter, two Braun-type UA pulverizers, small platform scales, and a table for rolling pulps.

The operating personnel of the drying room consists of a foreman on day shift, with three dryers and grinders; and lead men on afternoon and graveyard shifts, with two or three helpers. The drying room operates 24 hr per day for seven days a week.

As samples are brought in from the rigs, they are lined up in sequence and dried on the steam plates. Since there is such a large amount of sludge to be dried, the time of drying is more than an hour per sample, and may extend to 3 hr for exceptionally heavy, slimy samples. After drying, the sample is scraped off the table into a large pan and quartered by means of a Jones splitter. Three of these cuts are put in paper sacks; the fourth being set aside for pulverizing. Each of these cuts will weigh from 2½ to 3 lb. Paper tags are used, designating the sample number by hole as well as by footage. The cut which is pulverized and then thoroughly rolled is divided between five different pulp sacks holding from 150 to 200 gm each. One of these pulps is sent to the assay office of the St. Anthony Mining and Development Co. for field control.

The other four are sent to the Magma Copper Co. in Superior where one pulp is assayed directly for total and oxide copper; two are used in making up composites for total copper, oxide copper, gold, silver and molybdenum determinations. The fourth pulp is held in reserve.

Two of the three sacks of rejects are stored in tunnels at San Manuel, and one is stored at Superior. These are held in reserve for possible future examinations, or for further assay.

Records of the holes, listing each individual sample number, depth of footage, date of sample, weight of cut, number of splits made to obtain this cut, and the name of the sampler are kept in the bucking room. These records are all filed.

### ASSAYING

The pulp sent to the St. Anthony Mining and Development Co. is assayed for total and oxide copper. The method of assay used is a modified procedure of the short iodide method, wherein the end point of the starch-iodine blue was made much sharper, and gives results close enough for field control.

At the Magma Copper Co. the total copper content is determined by standard electrolytic methods; oxide copper is determined by the long iodide method. Results of these assays check very closely with those of Ledoux and Co. and Union Assay Office.

Gold, silver and molybdenum are determined from composites. These composites are made up of pulps representing portions of the hole of the same general mineralization or assay character, but do not usually exceed 100 ft of hole.

If the average of individual assays of total copper checks within 0.03 pct with the total copper determination of the composite by Magma Copper Co. and an umpire assayer, those values are accepted as being correct.

In the office at Superior, records of each hole are compiled showing all determinations made on individual and composite samples, the accepted assay, rock type, and mineralogy.

### TONNAGE AND GRADE

Churn drilling to May 1 has blocked out approximately 119,170,000 tons of ore, which will average 0.80 pct copper. This total contains 14,600,000 tons of low-grade oxidized material which averages 0.65 pct total copper, 0.36 pct oxide; 29,640,000 tons of mixed oxidized and sulphide ore which averages 0.82 pct total copper, and 0.58 pct oxide copper; and 74,930,000 tons of sulphide ore which averages 0.82 pct total copper, and 0.05 pct oxide copper.

These tonnage and grade figures are of a preliminary nature, and are kept up to date with the drilling as closely as the required checking of assays allows.

The following procedure is used to compute the tonnage and grade:

The drilling area is divided into rectangles and triangles as the spacing of drill holes demand, with a drill hole at each corner of the rectangles, and at the apices of each triangle. Care is taken to arrange the pattern of geometric figures so that one drill hole is not a common corner for more figures than necessary. The area, calculated from the horizontal distance between holes, is multiplied by the average of the thickness of the columns of ore encountered in the holes making up the corners of the figures. Twelve and one-half cu ft per ton is the factor used to convert the volume to tonnage. The grade for each block is the weighted average of the ore columns of the holes used.

The bottom sample, or in some cases more than one sample, is thrown out as a precaution against salting of marginal material by higher grade samples above. The cut back at the bottom of an ore column depends on the condition of the hole, grade of ore, and whether there is a gradational change in copper content at the bottom of the column, or a sharp break. The tonnage is calculated from drill-hole data with no allowances for an extension of the ore body beyond the boundaries of completed holes. This eliminates the necessity for constant revision of such an estimated extension of ore, because of the continually changing drilling pattern.

At intervals tonnage and grade calculations are computed by two other methods as checks. In the second method, the average area of adjacent cross sections is multiplied by the distance between the sections to compute the volume. The grade is computed from the weighted average of the ore columns of holes in the two sections. The ore columns are adjusted as in the first method.

The third method involves plotting the area of influence of each hole, and using the adjusted ore column for thickness, to compute a tonnage and grade figure for each area.

## ACKNOWLEDGMENT

Acknowledgment is due the officers of San Manuel Copper Corp. for permission to use the information presented.

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## DISCUSSION

(*B. S. Butler presiding*)

L. KENNETH WILSON*—Mr. Chairman, I am sure that most of us have been impressed with the authors' choice of a title: "San Manuel *Prospect*."

After being told here of some 119 million tons of ore reserves, I suppose that even a geologist will admit that is a pretty fair *prospect*.

The authors have called attention to two or three normal faults of which the San Manuel fault is the most prominent and mentioned that it is difficult to visualize a low angle fault of such magnitude. It has been suggested that this San Manuel fault was originally a steep structure and that it now appears flat because of tilting in the Black Hills.

The point I wish to submit for consideration here is that it appears to me that greater attention might be given to evidence of low angle thrusts and also of *horizontal* motion on fault planes.

By correlating the various information which is available my impression is that the regional picture is not entirely in harmony with the evidence reported on San Manuel, alone. I would suggest that the sequence of district geological events be defined by six episodes, as follows (Fig 4):

*Stage* 1. Under regional stress, compression fractures were opened and formed the northwest vein systems known as the Collins vein, the Mammoth vein—but *not* the Mammoth fault—the Mohawk vein and similar structures. Concurrently, during this earliest fracturing, low angle thrust faults were developed and I believe that the San Manuel fault is a conspicuous example.

*Stage* 2. This earliest fracturing was interrupted by northwest tear lines. I imagine that such tears would conform to the continental deformation which is best expressed in the Rocky Mountain province and in the Great Basin. These tears either prepared for or guided the emplacement of later intrusives—such as rhyolite, rhyolite breccia and related rocks.

*Stage* 3. The metals of the earliest surge of ore solutions then entered these same conduits. These metals include gold, silver, lead, zinc, molybdenum and vanadium, but only relatively slight amounts of copper.

*Stage* 4. Subsequent to this early mineral phase, a rejuvenation of easterly compression resulted in a thrust to the north of all fault blocks—for at least a distance of 500 ft measured along the strike of northwest faults. It is at this point that I feel we should interpret the maximum display of horizontal motion, and I believe this is a critical point to consider when we analyze the displacement on the so-called East fault, or the Mammoth fault.

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At this stage the Collins vein and the Mohawk vein are warped and buckled. The westerly dips of the Collins vein are steepened eastward, the steep easterly dips of the Mohawk vein are flattened still more to the east, and simultaneously tension breaks were developed in an east to northeast direction.

By "breaks" I do not mean only a simple fault plane but any zone of lineal rock-fracture. These are important. They are the so-called Turtle fault in the northern part of the District and another parallel structure one mile to the south—the structure which was to become the San Manuel ore body. There may be others.

*Stage* 5. Into this southern structure there was introduced the final phase of mineralization, a surge of copper solution which filled and finally obscured one of the more youthful elements of the structure pattern.

*Stage* 6. Lastly, the structural history is closed by the relief and sudden relaxation of this compressive stress which resulted in a general slump of all fault blocks. This youthful stage is marked by the conspicuous Mammoth Fault.

The Mammoth Fault has been thought by some to be a structure of great displacement (and perhaps even some economic significance) but I believe that the most recent displacement which we can read is only two or three hundred feet vertically.

And when we speak of greater vertical displacement I am afraid that we are neglecting to consider the effect of horizontal motion and of thrusting.

Eldred D. Wilson*—Steele and Rubly have suggested that, after oxidation and enrichment, the San Manuel fault was formed with a dip of 60 to 65° to the southwest, and that subsequently the region underwent tilting of 25 to 35° to the northwest.

This supposition of regional tilting implies some complications that seem worthy of question.

If Steele and Rubly's longitudinal section were tilted back 25 to 35° to the southwest, the known hypogene or primary ore body would then be a tabular pipe-like mass with steep plunge. Along its hanging wall there would be rather intense oxidation to depths of 2500 to 3000 ft below the old erosion surface, although some primary ore as shallow as 1300 ft remained unoxidized. Furthermore, the chalcocite zone under this set-up would range from 1100 to 2600 ft in depth. Is not such a pattern for extensive oxidation and enrichment rather unusual? Also, would not this interpretation imply that, prior to tilting the primary ore body farther south and southwest was too deep for notable supergene enrichment?

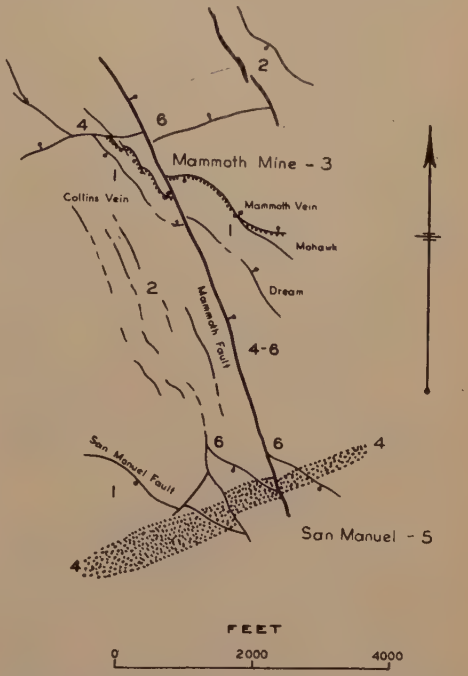

Fig 4—Structure pattern of the San Manuel Area. (*Wilson, after Schwartz and Peterson.*)

If the San Manuel fault originally dipped 60 to 65°, the Gila conglomerate on its hanging-wall side would necessarily exceed 4000 ft in thickness, which is considerably more than it is known to be elsewhere in the region.

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H. J. STEELE (author's reply)—The theory of tilting and faulting, as expressed in the paper, does imply complications, as stated by Dr. Eldred Wilson. We are however, confronted by the following facts established during our exploratory program:

1. The conglomerate, striking northwest is tilted to the northeast. The dip of the bedding varies from 25 to 45° with the average about 30°.

2. The contact of the conglomerate and the monzonite is a remarkably even plane striking more or less parallel to the strike of the conglomerate, but dipping to the southwest at 25 to 30°. By simple projection we have been able to predict the bottom of the conglomerate with accuracy as the drilling program has expanded. The drill cuttings at the contact indicate gouge and slickenside.

3. The bottom of oxidation and paralleling zone of secondary enrichment is dipping to the southeast at 40°. The dip of the ore body is 55 to 60°.

While no serious attempt has been made to work out the structure in detail, because we feel that churn drill data is inadequate for such a job, and time has not been available for a detailed surface geological study over a wide area; in order to provide a reasonable explanation for the above points it seems apparent that tilting and faulting must be considered. Two stages of tilting are required to explain the dip of the bottom of oxidation and the dip of the conglomerate. The deposit must have originally been in such a position that the bottom of oxidation was relatively horizontal and the chalcopyrite zone had an initial plunge to the southeast of 15 to 20°. Assuming that the bedding of the conglomerate approached the horizontal when it was deposited, it has obviously been tilted to its present position and the contact between monzonite and conglomerate has the physical aspects of a fault plane both at surface exposures and from drill cuttings. It can be argued that the conglomerate could have been deposited horizontally on an uneven erosional surface, and then tilted, but such a surface would have been abnormally steep and the top of the monzonite in the area covered by the San Manuel fault has none of the characteristics of a weathered surface.

There are many perplexing problems that will require a great deal of detailed study both surface and underground and I readily admit that we have proposed a simple and generalized explanation for a problem that is undoubtedly complex.

Mr. Kenneth Wilson, in suggesting a sequence of geological events, acknowledges the existence of the San Manuel fault and describes it as being a low angle thrust fault rather than a normal fault as we have interpreted it.

By his sequence of events he suggests that the San Manuel fault is premineral and that all mineralization in the district was post Gila in time. This does not seem probable as the felsite, which has been found to be post mineral both in surface exposures and drill cuttings, is cut by the San Manuel fault. We also have quite conclusive evidence that the most westerly extension of the ore body has been cut by the San Manuel fault. Mr. Wilson does not provide for the dip of the conglomerate, the bedding planes of which form an angle ot about 60° with the plane of the fault. This, in itself is one of the important bits of evidence for the existence of such a fault.

Drilling and outcrops in the northeast part of the deposit have quite definitely proved that the bottom of the conglomerate is in the footwall of the San Manuel fault and rests on the old erosion surface which slopes to the northeast at 15 to 20°. If this is correct, the fault could not be a thrust.

# Geology of the Castle Dome Copper Deposit, Arizona

By N. P. Peterson,* Member AIME

## Abstract

The Castle Dome copper deposit is of the porphyry type and occurs in a body of quartz monzonite intruded into the pre-Cambrian formations and possibly into the lower part of Paleozoic limestones. The hypogene sulphide minerals include pyrite, chalcopyrite, and a little molybdenite, sphalerite, and galena. Pyrite and chalcopyrite are disseminated or occur in small closely spaced veins. The hypogene mineralization shows distinct zoning in the distribution of pyrite and chalcopyrite, so that the mineralized area can be subdivided into pyrite and chalcopyrite zones. Supergene enrichment, though not extensive, played an important part in the formation of the ore body, the chalcopyrite zone being the protore. Several thin, gently dipping, diabase sills occur in the mineralized area, and the copper content of the rock in and near these sills is substantially higher than average. The formation of the ore body was caused by the combined effects of three forces: 1. Zoning of the hypogene mineralization, 2. Richer copper metallization associated with the diabase sills, and 3. Supergene enrichment.

## Introduction

The Castle Dome copper deposit is about 5 miles west of Miami, Arizona, and about 3 miles west of the Miami-Inspiration ore body. The Castle Dome open-pit mine is on the south flank of Porphyry Mountain, a prominent landmark of the region.

In 1941 the deposit was being explored by the Miami Copper Co., but the exploration program was cut short when the decision was made to develop the property as a war project. The Castle Dome Copper Co., Inc., was organized as a wholly owned subsidiary of the Miami Copper Co., and arrangements were made with Defense Plant Corp. for developing and equipping the mine and for construction of a concentrating plant. Construction and development work was begun in January 1942, and milling operations started June 10, 1943. Since that date, the mine has been in continuous operation by the Castle Dome Copper Co. under a lease agreement with Defense Plant Corp. producing copper at an average rate of 4,000,000 lb per month. To January 1, 1946 the total production amounted to 121,088,000 lb of copper, 4,271 oz of gold, and 211,755 oz of silver.

A detailed study of the copper deposit and the surrounding area was undertaken by the U. S. Geological Survey in June 1943. Other members of the Survey participating in the work were C. M. Gilbert, G. L. Quick, and J. P. Albers. The results of this work will be published as a comprehensive report by the Geological Survey.

## General Geology

The Globe quadrangle, which includes the Castle Dome area, was studied by F. L. Ransome[1] of the Geological Survey in 1901 and 1902. His maps and descriptions were

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Published by permission of the Director, U. S. Geological Survey. Presented at the Arizona Section meeting, Tuscon, Oct. 28–30, 1946. Manuscript received at the office of the Institute June 2, 1947. Issued as TP 2302 in Mining Technology, March 1948.

* Geologist, U. S. Geological Survey, Globe, Ariz.

[1] References are at the end of the paper.

published in Professional Papers 12 and 115.

Fig 1 is a generalized geologic map of the central part of the Castle Dome area. The columnar section of sedimentary volcanic rocks occurring within the map area is illustrated in Fig 2.

The Pinal schist, which is the oldest rock in the region, is a complex of metamorphosed sedimentary and igneous rocks of early pre-Cambrian age. Also of pre-Cambrian age but much younger than the Pinal schist are rocks of the Apache group. Though the two are nowhere in contact in this area, a major unconformity between them is indicated by the presence of schist

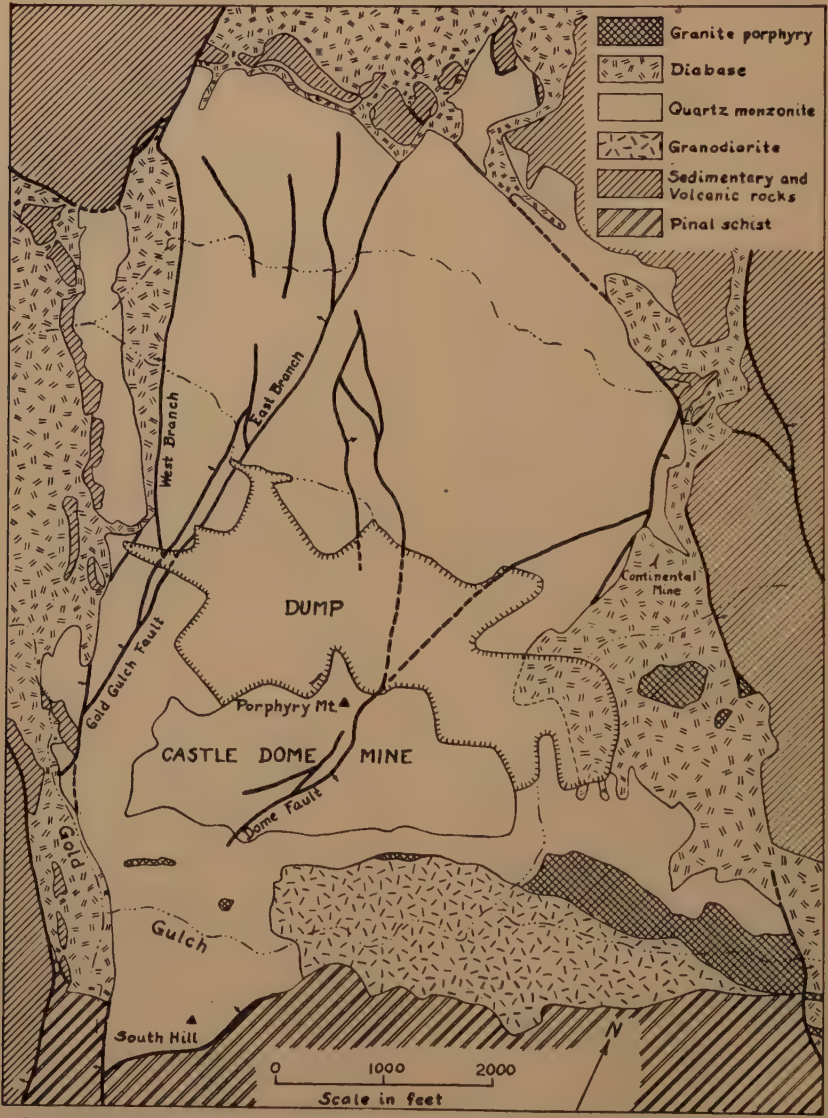

FIG 1—GENERALIZED GEOLOGIC MAP OF THE CENTRAL PART OF THE CASTLE DOME AREA, ARIZONA.

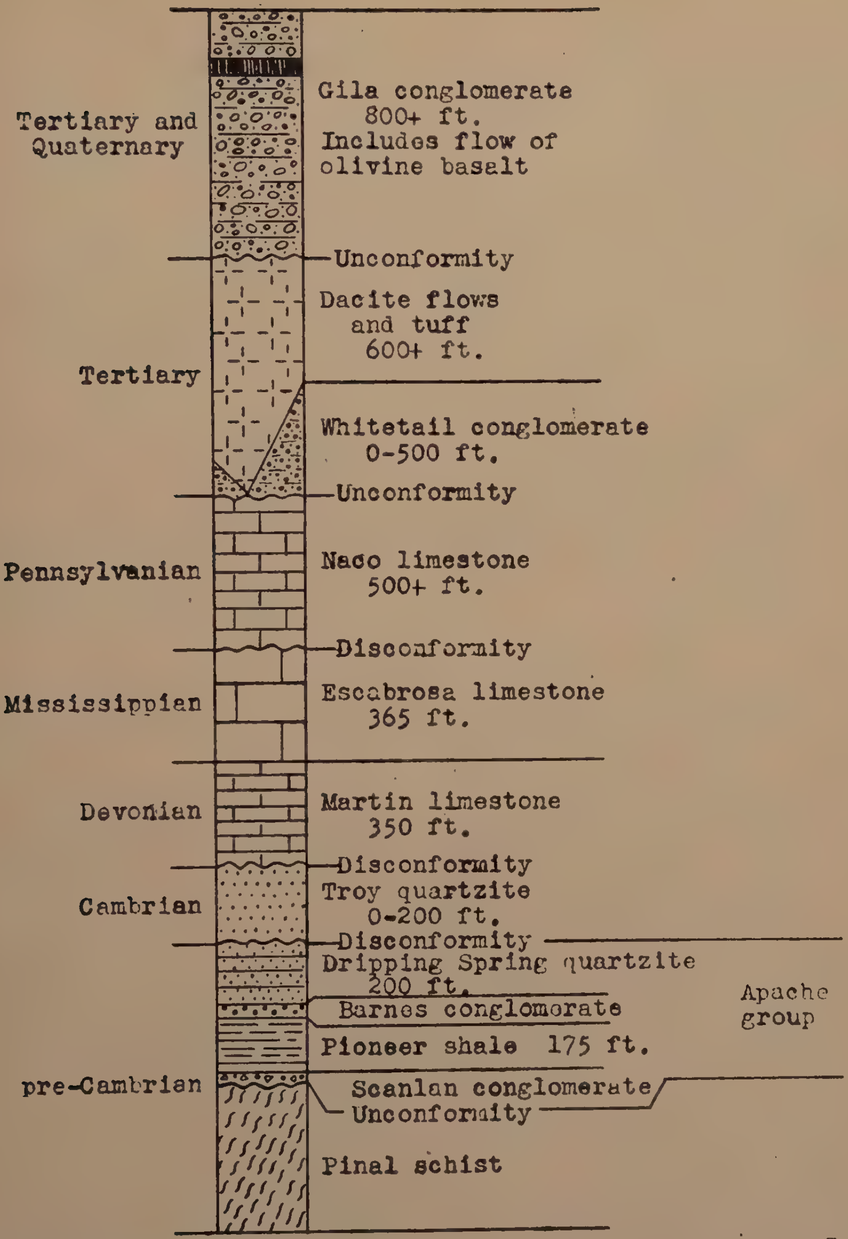

FIG 2—COLUMNAR SECTION OF SEDIMENTARY AND VOLCANIC ROCKS OCCURRING IN THE CASTLE DOME AREA.

pebbles in the Scanlan conglomerate, the basal formation of the Apache group.

The rocks of the Apache group and the Cambrian Troy quartzite were deeply eroded before the Devonian Martin limestone was deposited. The Mescal limestone (uppermost formation of the Apache group) and all but a few remnants of the Troy quartzite were removed by pre-Devonian erosion. Blocks of the Dripping Spring quartzite and Barnes conglomerate of the Apache group occur in the basal conglomerate of the Martin limestone. The Escabrosa limestone of Mississippian age is gradational with the underlying Martin limestone, and the contact between them is arbitrary. Between the Escabrosa limestone and the overlying Naco limestone of Pennsylvanian age there is no clear evidence of erosion, but the absence of upper Mississippian beds suggests a disconformity between them.

There is no record of sedimentation between the Pennsylvanian epoch and the deposition of the Tertiary Whitetail conglomerate. The history of that interval is one of faulting and the intrusion of a variety of igneous rocks. The first intrusion of this period was a small mass of biotite granodiorite, and it was followed by the larger biotite quartz monzonite complex in which the Castle Dome ore body occurs. The quartz monzonite intruded and metamorphosed the Scanlan conglomerate and the Pioneer shale, basal formations of the Apache group, but its relation to the Paleozoic limestones is not known. It is tentatively regarded as post-Pennsylvanian in age, but it might conceivably be post-Apache and pre-Devonian.

Normal faulting followed the emplacement of the quartz monzonite and produced an uplifted block trending generally north through the central part of the area (Fig 1). It is within this block that the quartz monzonite and the copper mineralization are now confined. This faulting was followed during the Mesozoic era by the intrusion of great volumes of diabase, particularly along the east, north, and west sides of the quartz monzonite block and into the surrounding rocks. The diabase forced its way between the beds of sedimentary rock as sills and occupied many of the faults. The last intrusion of this period is represented by dikes and small masses of granite porphyry south and east of the mine. They intrude the quartz monzonite and the adjacent diabase. The copper mineralization followed the intrusion of the granite porphyry, probably during the Laramide revolution.

Faulting and the intrusion of such great volumes of magma relatively close to the surface must have caused elevation of the area initiating a period of extensive erosion. By the time streams began to deposit the Whitetail conglomerate, some time during the first half of the Tertiary period, much of the Paleozoic limestone had been eroded away, and the Apache rocks and the diabase were exposed in many places. Whether or not the quartz monzonite and granite porphyry intrusions had been largely uncovered is uncertain, but it is possible that such extensive erosion as preceded the deposition of the Whitetail had actually removed the roof from the quartz monzonite mass. The great variation in the thickness of the conglomerate from place to place shows that the surface on which it was deposited had moderate relief. The Castle Dome area as a whole, however, must have been relatively low-lying compared with the surrounding region for within it the conglomerate is thickest and most widespread. Probably most if not all of this area was buried by the Whitetail conglomerate.

Then followed a widespread outpouring of silicic lava forming a great sheet of biotite dacite, at least 1000 ft thick in some places, which covered the entire area and buried the Whitetail conglomerate. Again normal faults disturbed the area, and erosion laid bare the pre-Tertiary rocks in the uplifted areas, notably the old quartz monzonite block. In the valleys on this erosion surface, streams deposited the Gila conglomerate during late Pliocene and Quaternary time. The conglomerate filled many of these old valleys and spread more thinly over much of the higher parts of the area. The region was again faulted, and erosion developed the present rugged topography leaving remnants of the Gila conglomerate in the higher parts of the area. A final rejuvenation has caused entrenchment of the present stream channels.

The rocks that are most important in their relationship to the copper deposit are the quartz monzonite, the granite porphyry, and some thin diabase sills intruded into the quartz monzonite in the vicinity of the mine. Although the granite porphyry is mineralized, it was only slightly affected by supergene enrichment and contains no ore bodies probably because it is less pervious to percolation of ground water. The granite porphyry may be an offshoot of the Schultze granite, the main body of which lies south of the Castle Dome area. Although the relative ages of the two rocks cannot be accurately determined, they are approximately contemporaneous. Both the thin diabase sills and the large diabase intrusions are later than the quartz monzonite and are pre-mineral age, but the two types of diabase bodies are nowhere in contact, and no evidence has been found to indicate that they are related intrusions.

The outcrop of the quartz monzonite mass measures about one and one-half miles from south to north by about one mile from east to west. Porphyry Mountain is in the south-central part of the outcrop. The quartz monzonite is a coarse-grained, porphyritic rock consisting essentially of quartz, orthoclase, oligoclase, and biotite. Two widespread textural varieties have been distinguished, but although they differ in texture, the mineral composition of the two is essentially the same. One variety is porphyritic quartz monzonite containing large phenocrysts of pink or reddish orthoclase in a coarse-grained groundmass, the other is quartz monzonite porphyry containing phenocrysts of orthoclase, quartz, oligoclase, and biotite in a relatively fine-grained groundmass. No appreciable difference in either mineralization or hydrothermal alteration is recognizable in adjacent bodies of the two phases. The quartz monzonite is much older than the copper mineralization which is more nearly contemporaneous with the granite porphyry and probably genetically related to it.

## STRUCTURE

The dominant structure of the Castle Dome area is a horst trending in a north-northwest direction through the central part of the area. Within it is the quartz monzonite. The horst is bounded on the east and west sides by steeply dipping normal fault systems along which the quartz monzonite was brought into fault contact with pre-Cambrian Apache and Paleozoic rocks that formed the roof of the quartz monzonite intrusion. Diabase was later intruded along these marginal faults and now generally separates the quartz monzonite from the relatively depressed blocks where the sedimentary rocks are exposed. A second and smaller horst trending nearly north is bounded on the west by the East Branch of the Gold Gulch fault, which dips west, and on the east by the east-dipping Dome fault system and a group of east-dipping faults north of the mine. A fault, occupied along part of its extent by diabase, crosses the main horst near the northern edge of the quartz monzonite, and along it the block to the south was relatively elevated.

At least three distinct periods of major faulting are recognized along the margins of the horst. The oldest faults displaced the Pennsylvanian limestones and the quartz monzonite but are older than the intrusion of the diabase. These displacements initiated the horst structure, undoubtedly during the Mesozoic era. Displacements probably as much as 1000 ft occurred along both sides of the horst at this time. The first displacement along the Dome fault also probably occurred at this time. Major faulting occurred again during the Tertiary period after the outpouring of the dacite and before the Gila conglomerate was deposited. The last major fault movements displaced the Gila conglomerate and occurred during the Quaternary period.

The most prominent structure in the mine area is the Dome fault system which consists of several related faults cutting northeastward across the central part of the ore body. The Dome fault appears to displace the main diabase sill in the mine at least 300 ft; however, part of the movement on the fault is prediabase, and may have displaced the fracture into which the diabase was injected. Other smaller faults roughly parallel to the Dome faults caused small offsets in the sills with the downthrow side always on the east.

A set of closely spaced fractures striking about N75°E and dipping steeply southward is now represented by small veins in the mineralized area. A few fractures having other attitudes are present, but they do not have any systematic arrangement with respect to vein fractures. A set of generally unmineralized shear joints and small faults striking from northwest to north-northwest and dipping moderately northeastward is also prominent; and east of the Dome fault, fractures having this attitude are very closely spaced. Attempts to relate the vein fractures to the local fault pattern have been unsatisfactory.

### The Hypogene Mineralization

The hypogene copper mineralization is confined almost entirely to the granite porphyry and to the southern half of the quartz monzonite body. Where it extends beyond the limits of these rocks it is usually very weak.

The principal hypogene sulphides in order of abundance are pyrite, chalcopyrite, and molybdenite. They occur mainly in and associated with a set of narrow, closely spaced, generally parallel quartz veins that strike N75°E and dip steeply southward. Many of the veins contain only quartz and pyrite, some contain quartz and chalcopyrite or quartz and molybdenite, and still others contain quartz with all three sulphides in various proportions. The only other hypogene sulphides, sphalerite and galena, appear to represent a decidedly later phase of mineralization and occur in a few veins along faults and major fractures that generally cut across the earlier copper veins. The gangue minerals are mainly the same as the constituents of the host rock, together with introduced quartz, clay, sericite, and a little barite.

The distribution and relative proportions of the hypogene sulphide minerals show a very distinct zoning pattern. In one zone, which passes through the summit of Porphyry Mountain and extends across the quartz monzonite in a direction parallel to the strike of the veins, the mineralization is mainly quartz and pyrite with very little chalcopyrite (Fig 3). This zone in which the rock contains only about 0.10 to 0.15 pct copper is referred to as the high-pyrite zone. Northward the mineralization in the high-pyrite zone gradually fades out with no apparent change in its character. Adjacent to this zone on the south and parallel to it, is a zone in which there is less pyrite and more chalcopyrite; the copper content in the form of hypogene minerals is about 0.3 pct in this zone. It is referred to as the chalcopyrite zone and is the protore of the ore body.

A much smaller zone of high-pyrite mineralization occurs along the north side of the fault that forms the contact between the quartz monzonite and the schist on the south side of South Hill (Fig 3). Adjacent to this small high-pyrite zone on the north is a small chalcopyrite zone in which supergene enrichment has produced a small ore body probably comparable in grade to the main ore body. Between the two chalcopyrite zones is a weakly mineralized belt in which no ore has been found.

Molybdenite is most abundant along the southern margin of the main high-pyrite zone and in the northern part of the chalcopyrite zone. In other words, it could be said to occupy an intermedite position in the zoning pattern between pyrite and chalcopyrite. The trend of the various

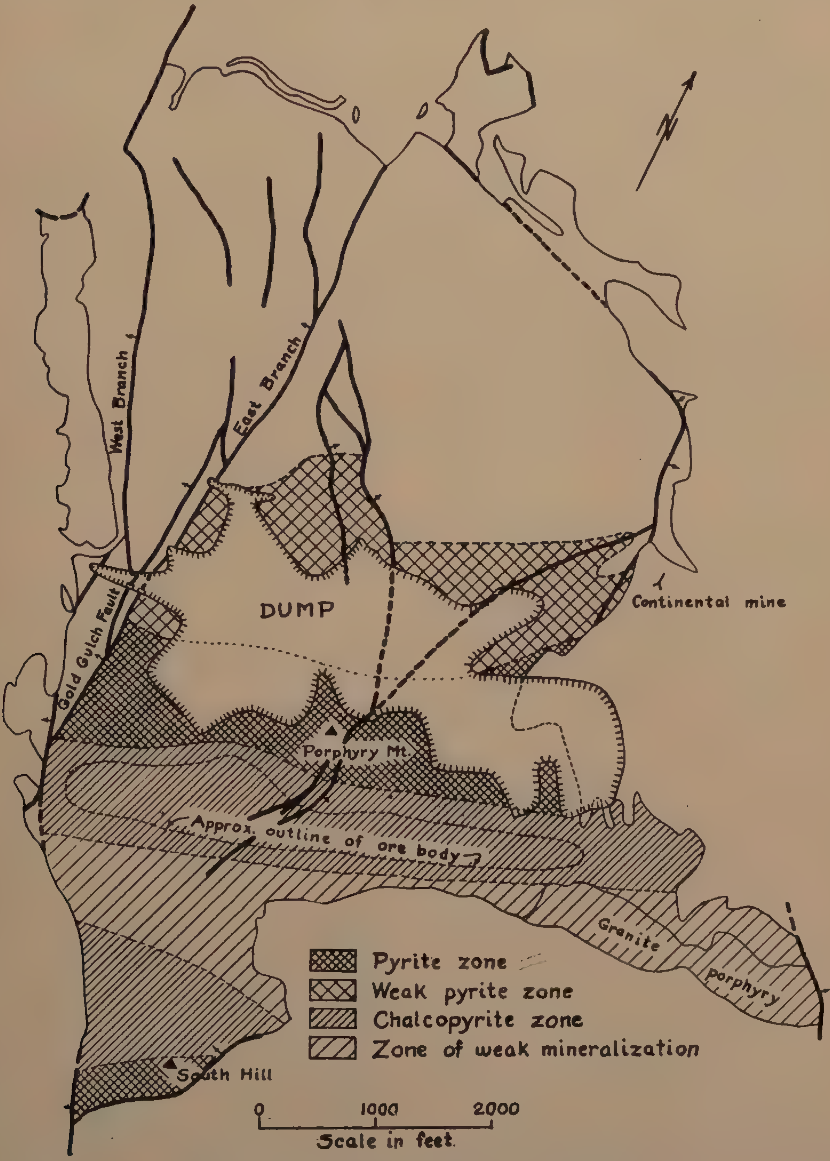

FIG 3—OUTLINE OF THE MAIN MASS OF QUARTZ MONZONITE SHOWING GENERALIZED PATTERN OF METALLIZATION ZONING.

zones is roughly parallel to the strike of the veins, and in dip the zones are probably also parallel to the veins.

### Hydrothermal Alteration of the Host Rock

The hydrothermal alteration of the host rock[2] consists of three phases: a weak propylitic phase, a clay phase, and a quartz-sericite phase. In general, the alteration appears feeble compared with that in most other disseminated copper deposits. Except in a few places where the quartz-sericite phase is intense, the original texture of the quartz monzonite is not obscured by alteration; in most places the general appearance of the rock is little changed. The areal distribution of the various phases and their relative intensities are clearly related to the zoning pattern of the hypogene metallization.

Very weak alteration of the propylitic type, in which biotite and plagioclase are partly replaced by sericite, epidote, clinozoisite, chlorite, calcite, and a little pyrite, occurs in an outer zone where metallization is also very weak. The propylitic alteration extends somewhat beyond where metallization and alteration can be recognized in the field.

This feeble zone surrounds an area of stronger alteration where most of the plagioclase and a little of the orthoclase and biotite are replaced by a mixture of montmorillonite-type clay, hydrous mica, and sericite. The clay phase is easily recognized by the chalky appearance of the plagioclase which is commonly slightly stained by copper and iron compounds. The altered plagioclase slakes quickly, and freshly broken rock crumbles when wet because of the swelling of the clay. The clay alteration is most intense in the high-pyrite and chalcopyrite zones and diminishes gradually toward the north and south where more fresh plagioclase together with alteration products of the propylitic phase is evident.

The third phase of alteration, referred to as the quartz-sericite phase, is related to numerous small quartz-pyrite veins along which the wall rock is replaced by quartz, sericite, and a little pyrite and adularia. Each quartz-pyrite vein is bordered by a glistening white zone in which the wall-rock has been mostly replaced by quartz and sericite. This type of alteration is general throughout the mineralized area but is most intense in the high-pyrite zone where pyrite veins are largest and most numerous. In some places, the alteration borders along the veins coalesce, and the rock is completely replaced. The quartz-sericite phase might be considered as forming an innermost zone of alteration coinciding with the zone of high-pyrite metallization, although it is actually superimposed on the clay and propylitic phases wherever the pyrite veins occur in them. The clay alteration is most obvious in the chalcopyrite zone where there are fewer pyrite veins.

### Structural Control

Very little is known concerning the structural controls of the hypogene mineralization. The structural feature that exerted the most obvious control over the mineralization was the set of closely spaced, nearly parallel fractures that are now represented by the veins. These fractures, however, were probably only secondary conduits by which the solutions were disseminated throughout the mineralized area. The main feeder channels, through which the solutions were conducted from their source to the minor fractures have not been recognized. The location of the most intense metallization and alteration centered near the summit of Porphyry Mountain, that is, in the high-pyrite zone, suggests that the main conduit lies in that vicinity, possibly a major fault parallel to the veins; but if such a structure actually exists, it is concealed by dumps and no direct evidence of its presence has been found.

Several major faults controlled the course of the mineralizing solutions to some extent. For example, the alteration along the Dome fault system is perceptibly stronger than average, and in the northern part of the area mineralization extends farther northward along one or both sides of some of the larger faults than elsewhere. Also the mineralization fades abruptly at the west along the big Gold Gulch fault and is very weak west of the fault (Fig 3).

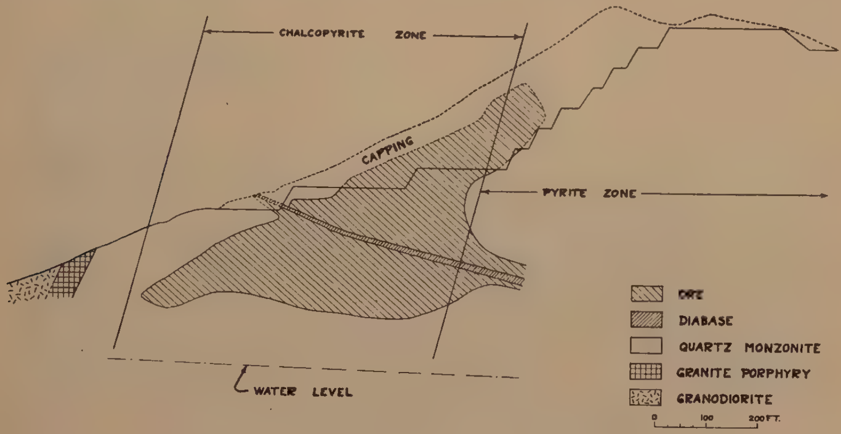

FIG 4—TYPICAL CROSS-SECTION OF THE ORE BODY.

### CONTROL BY THE DIABASE SILLS

Several thin diabase sills from 1 to 10 ft-thick (on Fig 4, but too small to be shown on Fig 3) crop out along the southern edge of the chalcopyrite zone and dip at low angles toward the north or northwest through the chalcopyrite and pyrite zones. The diabase in these sills is thoroughly brecciated and strongly mineralized. The mineralized veins, which dip approximately at right angles to the dip of the sills, do not continue through them, but abundant pyrite and chalcopyrite were deposited in random fractures in the brecciated diabase. The copper content of the sills is usually at least double that of the zone in which the sills occur, and the quartz monzonite adjacent to one or both sides of the sills is substantially higher than average grade. The stronger copper metallization associated with these diabase sills was an important contributing factor to the formation of the ore body.

### SUPERGENE ENRICHMENT

Supergene enrichment was also an important contributing factor in the formation of the ore body, although the enrichment cycle is in an early stage of development as is shown by the incomplete replacement of the hypogene sulphides by chalcocite and covellite and by the comparatively shallow depth to which enrichment has penetrated. In general, supergene enrichment has affected only the upper part of the ore body. It is relatively deep in the southern part but becomes progressively shallower and weaker toward the north. The leached capping, which averages about 80 ft in thickness, generally carries less than 0.1 pct copper; and the transition from waste to ore, as shown by the drill logs, usually occurs in less than 15 ft. The boundary between capping and the top of the ore conforms very closely with the present topography. The bottom surface of the secondary sulphide zone is much more irregular and apparently was not influenced by the ground-water level which is well below the ore body and about at the level of Gold Gulch south of the mine.

In the upper part of the secondary sulphide zone, most of the chalcopyrite has been replaced by chalcocite and covellite; but even here, pyrite is not replaced except to a very minor degree in a few places. The amount of replaced hypogene sulphide decreases progressively with depth. In many places, particularly near the footwall and in the western part of the ore body, unenriched hypogene ore is being mined. It is ore mainly because of the richer hypogene metallization associated with the diabase sills.

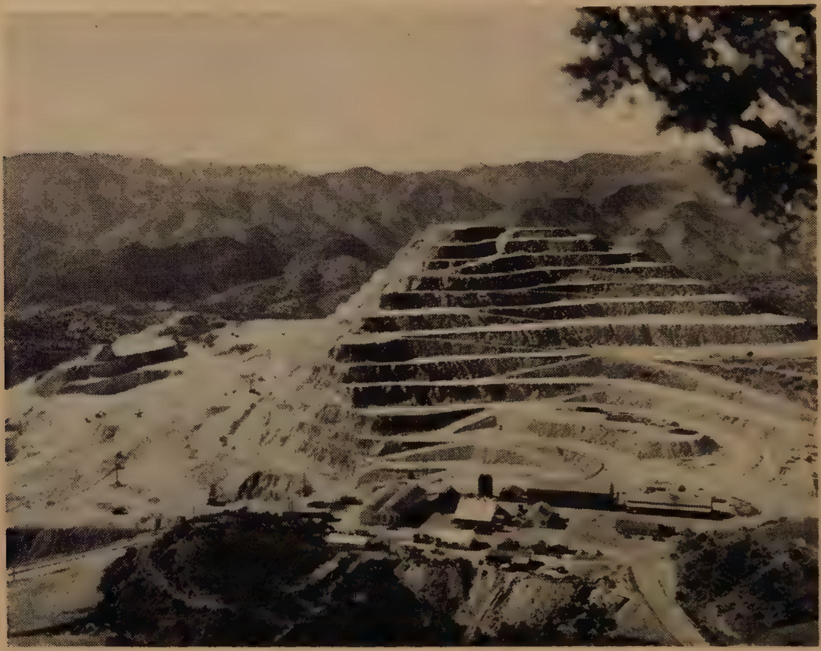

FIG 5—VIEW OF CASTLE DOME MINE LOOKING WESTWARD.
The mine is on the south slope of Porphyry Mountain; the north slope is covered by waste dumps. Photo by Kelley's Studio, Miami.

An interesting feature in connection with the supergene enrichment is the high concentration of covellite in the upper part of the secondary sulphide zone. Undoubtedly some covellite formed by direct replacement of chalcopyrite, but the greater part of it formed by oxidation of chalcocite. Evidence at Castle Dome seems to indicate that the enrichment process operated well above the permanent water table. At the start of the cycle, pyrite and chalcopyrite were broken down by near-surface weathering processes, and the copper thus set free was carried downward in solution until it came in contact with enough chalcopyrite to precipitate it as chalcocite. As oxidation and leaching continued to penetrate deeper, they reached the newly deposited chalcocite which was then broken down and its copper carried a little deeper. The first step in the leaching of chalcocite by ferric sulphate solutions is the removal of one half of its copper thus forming covellite which is much more resistant to oxidation than chalcocite. In this way a layer rich in covellite was formed at the top of the chalcocite zone.

### THE ORE BODY

The Castle Dome ore body is not simply a flat chalcocite blanket. Its length ranges from about 2000 ft on the upper levels to about 3800 ft on the lowest mine level. In cross-sections, the ore body is generally roughly triangular in shape with one vertex of the triangle at the apex of the body (Fig 4). The upper or south side is the boundary between capping and the top of the secondary sulphide zone. It dips southward roughly parallel to the south slope of Porphyry Mountain. The north or footwall side coincides approximately with the north limit of the chalcopyrite zone. In detail the shape of the ore body is influenced by the diabase sills and to some extent by faults. Most of the irregularities in the upper surface of the ore were caused by faults, in some cases by deep leaching along them, in other cases by diverting the flow of ground water so as to insulate certain parts from the effects of supergene enrichment.

The displacement along the Dome fault system was partly premineral and partly postprimary mineralization; but since the boundaries of the ore body are largely the result of later supergene enrichment, the displacement had little effect on the continuity of the ore body.

## GEOLOGIC CONTROLS THAT PRODUCED THE ORE BODY

The formation of the ore body was due to three major causes: 1. Zoning in the hypogene mineralization, 2. The richer than average copper mineralization associated with the diabase sills, and 3. Supergene enrichment. No one of these causes was far reaching enough in its effects to produce an ore body large enough to be exploited by the large-scale methods employed in the present operation. However, by an accident of nature the effects of all three causes were superimposed, and by their combined effects they produced the Castle Dome ore body.

The first contributing factor was the result of primary zoning which caused a concentration of copper in a relatively small part of the mineralized area, that is, in the chalcopyrite zone. This in itself would have given the rock a copper content of only about 0.3 pct. However, in and adjacent to the diabase sills where they cut through the chalcopyrite zone, the hypogene mineralization was at least twice as rich in copper as the rest of the chalcopyrite zone.

When erosion cut through the overlying younger rocks, it exposed the chalcopyrite zone so that the diabase sills crop out along its southern limit, and thus not only all the richer protore associated with the diabase was preserved but also the southern part of it was brought within the range of supergene enrichment. The mineralized quartz monzonite was probably first exposed where Gold Gulch cut through the younger rocks, and therefore supergene processes have been active for a longer period of time in the vicinity of Gold Gulch than elsewhere in the area.

The topography was carved by erosion in such a way that the maximum amount of supergene enrichment occurred in the chalcopyrite zone so as to produce a chalcocite blanket directly overlying the rich hypogene ore associated with the diabase sills. If erosion had been a little deeper, the rich hypogene ore associated with the diabase would have been carried away and lost. If it had been shallower, the ore body would probably have been diluted by a layer of low-grade rock separating the supergene ore from the rich hypogene ore in the diabase. If Gold Gulch had not established itself exactly where it did, supergene processes might not have operated effectively enough to form an enriched ore body worth developing under existing conditions.

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2. Peterson, N. P., Gilbert, C. M., and Quick, G. L.: Hydrothermal Alterations in the Castle Dome Copper Deposit, Arizona. *Econ. Geology* (1946) **41**, 820–840.

# The Story of Eureka

By William Sharp, Member AIME

(Chicago Meeting, February 1946)

Discovery of new ore in the Eureka district, Nevada, as a result of bold and persistent exploration based on a geologic interpretation of structure has recently aroused wide comment in mining circles. It may therefore be of interest to the profession to have a summary of the historical background and an account of the recent program of development.

The mines of Eureka, discovered in the late 1860's, were the first important silver-lead mines in the United States. They have a production record of more than fifty-one million dollars.[1]

After the ore bodies were apparently bottomed in the 1880's, repeated attempts to find new ore were unsuccessful until 1940, when a diamond-drill hole cut a 40-ft width of sulphide ore 1409 ft below the 841-ft level of the Locan shaft. The next hole cut 30 ft, of ore and three additional holes drilled in 1943 and 1944 showed the ore body to be continuous, Shaft sinking to reach and open up the ore body has been in progress since 1945.

## Geology

### *Stratigraphy*

The ore in the Eureka district occurs in a series of Cambrian sediments which is exposed through a thickness of over 6000 ft. The stratigraphic column is as follows:[2]

| | Thickness, Ft |
|---|---|
| Ordovician: | |
| Pogonip limestone (upper part). Thin bedded limestone with some chert. | |
| Upper Cambrian: | |
| Pogonip limestone (lower part). Thin-bedded limestone (with shaly limestone and chert).... | 425+ |
| Dunderberg shale. Shale (clay and calcareous shale) and thin-bedded limestone............. | 340+ |
| Hamburg dolomite. Dolomite and limestone.................... | 900+ |
| Middle Cambrian: | |
| Secret Canyon shale. Shale and thin-bedded limestone........ | 1,035 |
| Geddes limestone. Carbonaceous, flaggy limestone with some shale partings................ | 335 |
| Middle (?) and Lower Cambrian: | |
| Eldorado dolomite. Dolomite and limestone.................... | 2,000 |
| Lower Cambrian: | |
| Prospect Mountain quartzite. Quartzite with occasional intercalated conglomerate and shale | 1,660+ |
| The Prospect Mountain quartzite has been subdivided into two formations: Upper part, Pioche shale. Shale, quartzite with some intercalated limestone (Lower Cambrian fossils)-200 ft. Lower part, Prospect Mountain quartzite proper- | 1,460+ft. |
| Total thickness................ | 6,695+ |

Much the most productive of these formations has been the Eldorado dolomite, which yielded more than $42,000,000 at Ruby Hill and an additional amount

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Manuscript received at the office of the Institute Sept. 6, 1946. Issued as TP 2196 in Mining Technology, September 1947.

* Geologist, Eureka, Nevada.

[1] B. F. Couch and J. A. Carpenter: Nevada's Metal and Mineral Production. Nevada *Bulletin* (1943) 37.

[2] H. E. Wheeler and D. M. Lemmon: Univ. Nevada *Bulletin* (May 1, 1939) 33.

from south of Ruby Hill, where, together with other formations (the Secret Canyon, Hamburg and Dunderberg) it produced nearly $7,000,000. The production from immediately north of Ruby Hill, in the Hamburg, Dunderberg and Pogonip formations, amounted to slightly more than $2,000,000.

### *Structure*

The present complex structure of Ruby Hill represents an anticline which was broken by thrust-faulting and later by normal faults.

*Folding and Thrusting*—The original fold was a northerly trending anticline with isoclinal limbs. Thrusting, apparently from the west, cut through the anticline, overturned the east limb and overrode it. The sedimentary beds in the overthrust block were in certain places materially thinned by slipping along approximately parallel planes, which can be seen within the formations. As a result, the Secret Canyon shale and the Eldorado dolomite were found to be less than half their normal thickness where penetrated by diamond drilling from the 841-ft level of the Locan shaft, and the Geddes limestone was completely missing in hole *C* as shown in section *A–A′* of Fig 1.

*Normal Faulting*—After the overthrusting, normal displacements occurred along faults of three sets, with strikes respectively west of north, east of north and nearly east-west. The effect of the east-west faults was to elevate the blocks to the south of them. This permitted erosion to remove the greater part of the overthrust beds on Prospect Mountain, where only remnants and down-dropped segments now remain, and to expose the overturned east limb of the anticline. At Ruby Hill and to the north on Adams Hill, however, all of the overthrust Cambrian sediments remain, and it is in these beds that the ore in this part of the district occurs.

### *Ore Bodies*

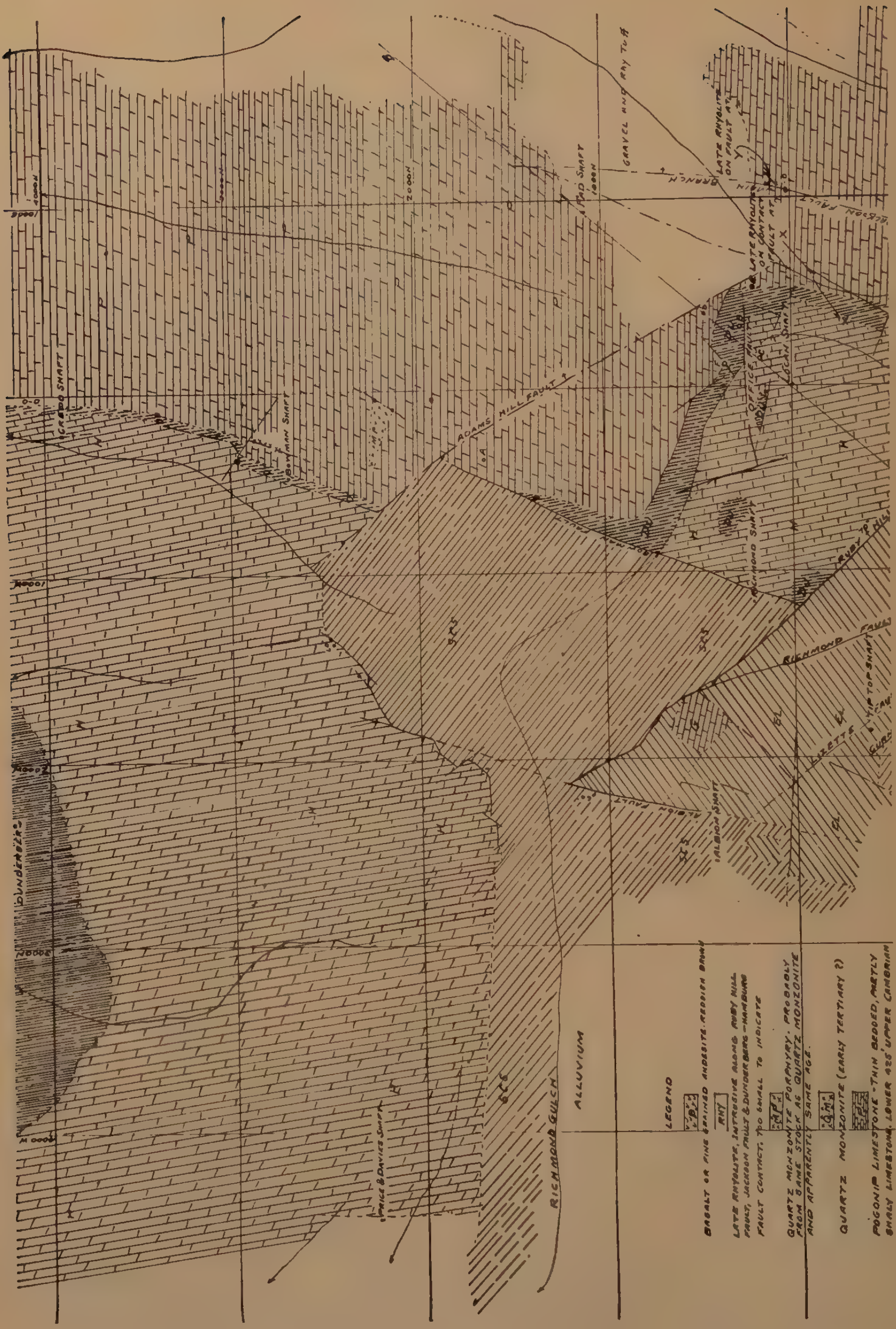
The ore bodies at Ruby Hill were all within a block of Eldorado dolomite, which narrows downward in the form of a wedge. It is bounded on the northeast by the Ruby Hill fault (one of the northwesterly striking normal set) and on the southwest by Prospect Mountain quartzite (which underlies the Eldorado). The quartzite dips eastward toward the fault, thus forming the bottom of the productive block.

The block, elongated in a northwesterly direction, has a length of 4400 ft on the surface. In width it measures 120 ft near its southeast end on the Jackson property and 1200 ft at its northwest end on the Richmond property. Downward, it came to a wedge end between the quartzite and the fault at a depth of 600 ft in the Jackson mine and at 1300 ft in the Richmond mine.

This remarkably productive wedge of dolomite has been crushed and recemented and into it the ore has been introduced. All the ore in this block was oxidized with the exception of occasional residual bunches in the cores of ore bodies. Although much of the lead was in the form of plumbojarosite, oxidation had not reached its extreme stage, since small amounts of anglesite and galena were present and most of the iron was carbonate.

An example of the richness of the ores mined by the companies in the early period can be gained from a published report of the Richmond Mining Co. for the year 1878. In that year, the average value of the ore smelted at its furnaces was given as follows: lead 33.12 pct, iron 24.07 pct, zinc 1.89 pct, arsenic 4.13 pct, sulphur 1.67 pct, together with a silver content of 27.55 oz per ton and a gold content of 1.59 oz per ton.

In mining this grade of ore, the companies had, of necessity, left casings of iron carbonate ore around the walls of all the stopes. Sometimes these casings were many feet thick. All the iron carbonate, con-


LEGEND
B
BASALT OR FINE GRAINED ANDESITE. REDDISH BROWN
RHY
LATE RHYOLITE, INTRUSIVE ALONG RUBY HILL FAULT, JACKSON FAULT & DUNDERBERG-HAMBURG FAULT CONTACT. TOO SMALL TO INDICATE
QMP
QUARTZ MONZONITE PORPHYRY. PROBABLY FROM SAME STOCK AS QUARTZ MONZONITE AND APPARENTLY SAME AGE.
QM
QUARTZ MONZONITE (EARLY TERTIARY ?)
P
POGONIP LIMESTONE - THIN BEDDED, PARTLY SHALY LIMESTONE. LOWER 425' UPPER CAMBRIAN
ALLUVIUM
RICHMOND GULCH
GRAVEL AND RAY TUFF
ADAMS HILL FAULT
RICHMOND FAULT
RUBY HILL
ALBION FAULT
ALBION SHAFT
RICHMOND SHAFT
LOCAN SHAFT
OFFICE FAULT
LATE RHYOLITE ON FAULT AT Y
LATE RHYOLITE ON CONTACT FAULT AT X
BRANCH
JACKSON FAULT
LIZETTE
HILLTOP SHAFT
DUNDERBERG
CRESO SHAFT
BOWMAN SHAFT
PRICE & DAVIES SHAFT
A MAD SHAFT
2000N
1000N
4000N
1000S
1000W
4000W
5000W
SCS
QL
P

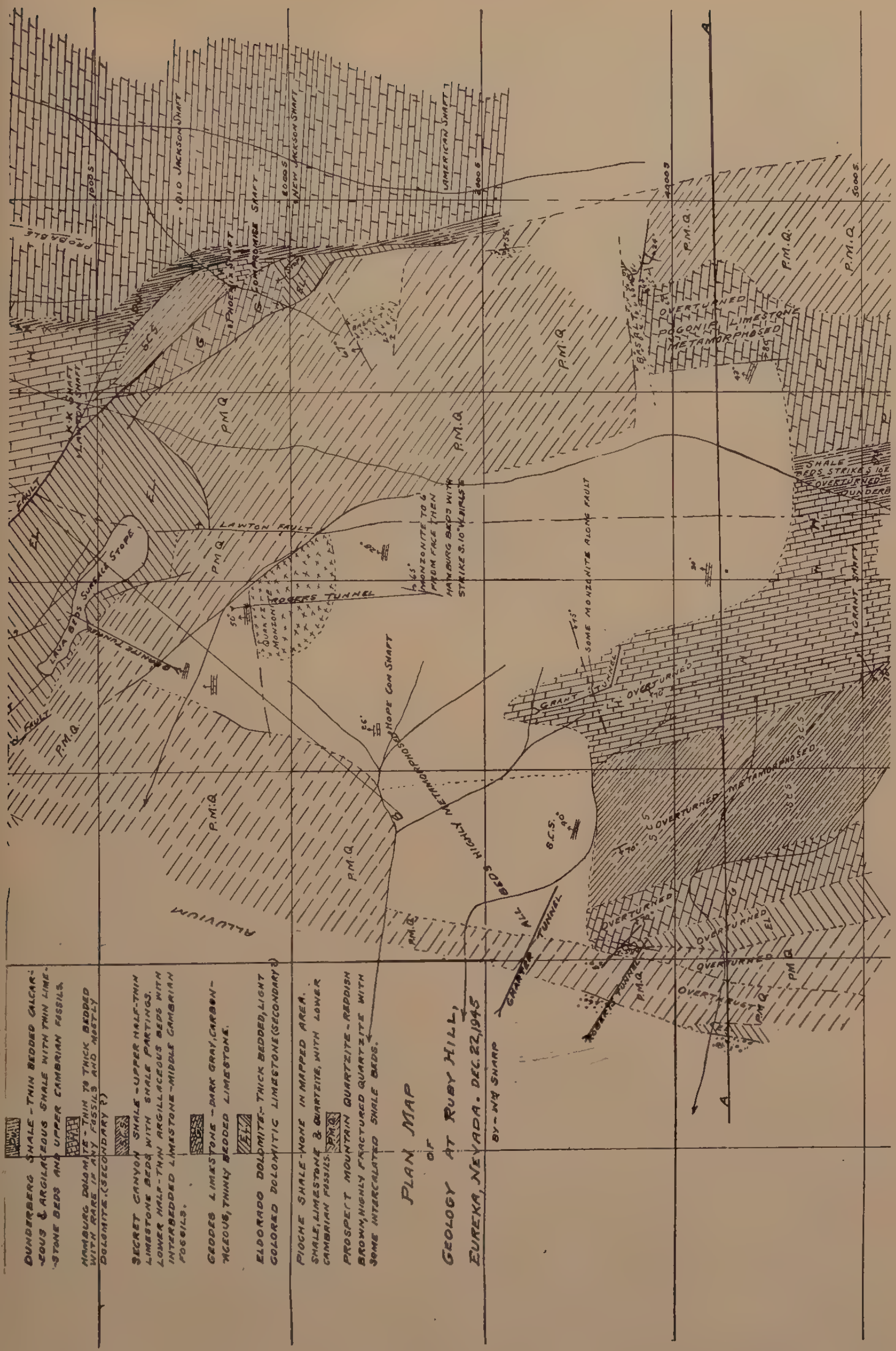
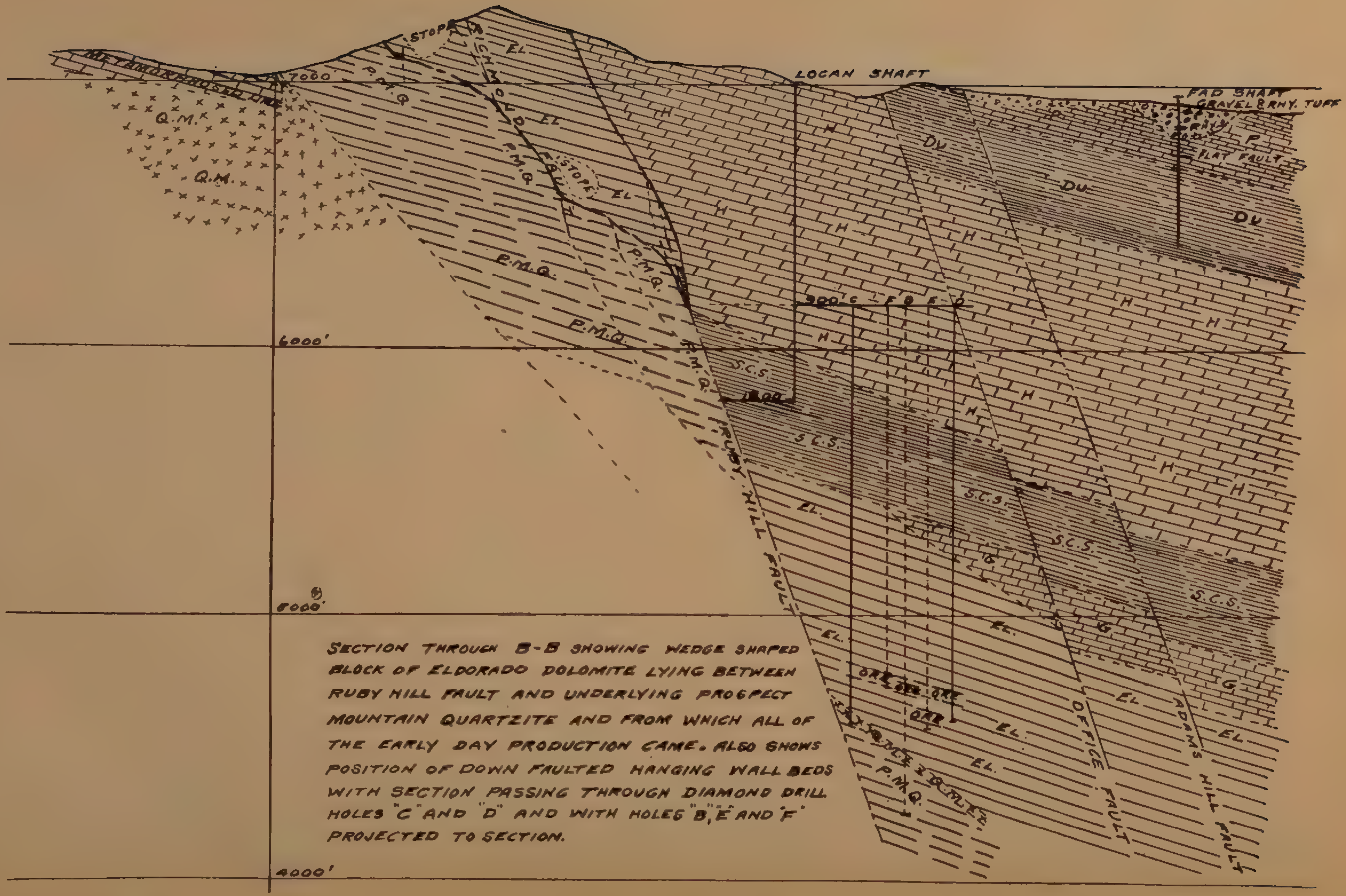

DUNDERBERG SHALE - THIN BEDDED CALCAREOUS & ARGILLACEOUS SHALE WITH THIN LIMESTONE BEDS AND UPPER CAMBRIAN FOSSILS.
HAMBURG DOLOMITE - THIN TO THICK BEDDED WITH RARE IF ANY FOSSILS AND MOSTLY DOLOMITE. (SECONDARY ?)
SECRET CANYON SHALE - UPPER HALF - THIN LIMESTONE BEDS WITH SHALE PARTINGS. LOWER HALF - THIN ARGILLACEOUS BEDS WITH INTERBEDDED LIMESTONE - MIDDLE CAMBRIAN FOSSILS.
GEDDES LIMESTONE - DARK GRAY, CARBONACEOUS, THINLY BEDDED LIMESTONE.
ELDORADO DOLOMITE - THICK BEDDED, LIGHT COLORED DOLOMITIC LIMESTONE (SECONDARY ?)
PIOCHE SHALE - NONE IN MAPPED AREA. SHALE, LIMESTONE & QUARTZITE, WITH LOWER CAMBRIAN FOSSILS.
PROSPECT MOUNTAIN QUARTZITE - REDDISH BROWN, HIGHLY FRACTURED QUARTZITE WITH SOME INTERCALATED SHALE BEDS.
PLAN MAP OF GEOLOGY AT RUBY HILL, EUREKA, NEVADA. DEC. 22, 1945
BY - W.A. SHARP
D
H
S.C.S.
G
EL
P.M.Q.
ALLUVIUM
OLD JACKSON SHAFT
NEW JACKSON SHAFT
AMERICAN SHAFT
PHOENIX SHAFT
COMPROMISE SHAFT
K-K SHAFT
PROBABLE
FAULT
LAWTON FAULT
SURFACE STOPE
LAVA BEDS
ROGERS TUNNEL
MONZONITE TO 6' FROM FACE THEN HAMBURG BEDS WITH STRIKE S.10°W. DIP 15°
QUARTZ MONZONITE
HOPE CON SHAFT
HIGHLY METAMORPHOSED
SOME MONZONITE ALONG FAULT
GRANT TUNNEL
GRANT SHAFT
OVERTURNED
OVERTURNED POGONIP LIMESTONE METAMORPHOSED
SHALE BEDS STRIKE S.10°E. OVERTURNED DUNDERBERG
OVERTURNED METAMORPHOSED
OVERTHRUST
CHARTER TUNNEL
ALL BEDS
1000 S.
2000 S.
3000 S.
4000 S.
5000 S.
A
A

FIG 1—GEOLOGIC STRUCTURE OF EUREKA DISTRICT.



SECTION THROUGH B-B SHOWING WEDGE SHAPED BLOCK OF ELDORADO DOLOMITE LYING BETWEEN RUBY HILL FAULT AND UNDERLYING PROSPECT MOUNTAIN QUARTZITE AND FROM WHICH ALL OF THE EARLY DAY PRODUCTION CAME. ALSO SHOWS POSITION OF DOWN FAULTED HANGING WALL BEDS WITH SECTION PASSING THROUGH DIAMOND DRILL HOLES "C" AND "D" AND WITH HOLES "B", "E" AND "F" PROJECTED TO SECTION.

tained small quantities of lead, gold and silver but within the casings there were also streaks and bunches of the high-grade ores. It was from these casings that production by lessees came when company operations ceased.

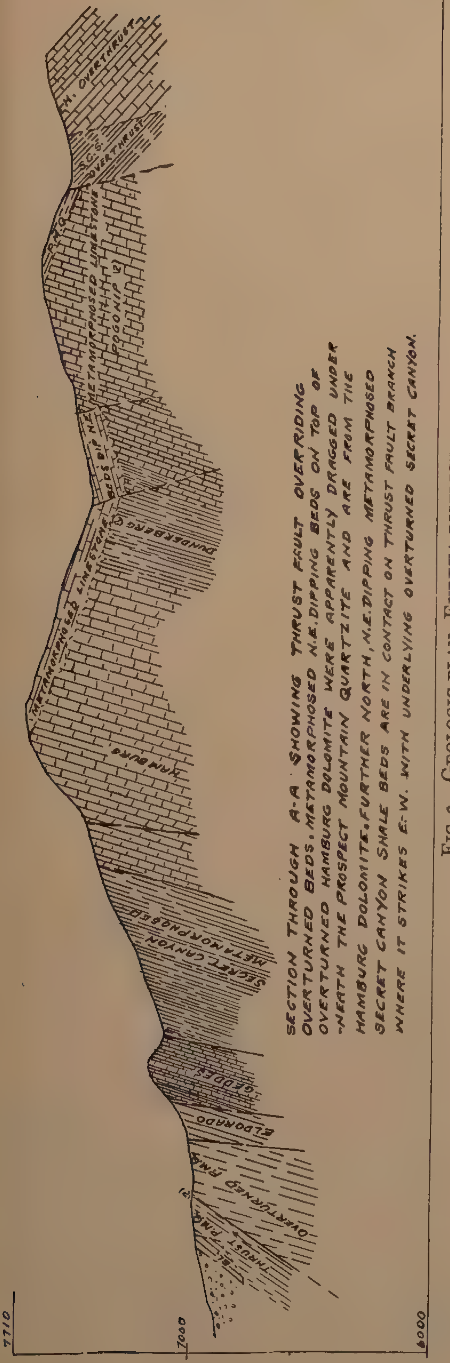
Most of the important ore bodies had no outcrops, although discovery of the mines was made on the Eureka Consolidated property where a large ore body outcropped in the dolomite near its contact with the underlying quartzite. Smaller surface lenses also made in the Jackson and Phoenix mines near this same contact but the only outcrop of ore in the Richmond mine was a narrow 40-ft lens that made along the Lizette fault. It was here that the Richmond company started its first shaft, the Tip Top incline. The ore in this shaft led to a large flat ore body a short distance below the surface, which made along a thrust-fault plane in the dolomite. A number of additional ore bodies along faulted segments of thrust planes were discovered in depth. Other types of ore bodies included:

1. Ore lenses along the Richmond fault, particularly where the fault separated the dolomite and quartzite.

2. Small ore bodies in and adjoining the Lawton cross fault.

3. Ore bodies along steeply dipping faults parallel to the Ruby Hill. On one of these a large ore body, known as the Potts Chamber, reached the bottom of the dolomite wedge at its junction with the Ruby Hill fault. Here the ore ended at a depth of 60 ft below the 800-ft level of the Richmond mine. Further to the northwest some stopes in this mine reached a depth of 900 ft but no ore of any consequence was found below that level.

The normal displacement on the Ruby Hill fault had dropped younger sediments on the northeast or hanging-wall side against the productive Eldorado dolomite. Clearly the most promising place to look for new ore was at a deeper stratigraphic



SECTION THROUGH A-A SHOWING THRUST FAULT OVERRIDING OVERTURNED BEDS. METAMORPHOSED N.E. DIPPING BEDS ON TOP OF OVERTURNED HAMBURG DOLOMITE WERE APPARENTLY DRAGGED UNDERNEATH THE PROSPECT MOUNTAIN QUARTZITE AND ARE FROM THE HAMBURG DOLOMITE. FURTHER NORTH, N.E. DIPPING METAMORPHOSED SECRET CANYON SHALE BEDS ARE IN CONTACT ON THRUST FAULT BRANCH WHERE IT STRIKES E-W. WITH UNDERLYING OVERTURNED SECRET CANYON.

FIG 2—GEOLOGIC PLAN, EUREKA DISTRICT.

horizon on the hanging-wall side below these younger sediments. Workings to test this horizon succeeded in reaching the dolomite but did not explore it very extensively. Geologists in the eighties disagreed as to whether the Ruby Hill fault was premineral or postmineral, and the mining companies, hoping that it was postmineral but fearing that it was premineral, made only half-hearted attempts to meet the huge costs of this deeper exploration.

Undoubtedly the fault is premineral, and acted as a channel for the ore-bearing solutions to reach the wedge of Eldorado dolomite, but the early operators feared that if it were premineral no ore could be expected in the hanging-wall formations.

This was the situation when active operations ceased.

## EARLY PRODUCTIVE PERIOD

In the early 1870's, Eureka boasted of two large smelters with a daily capacity of 200 tons or more each, and there were at one time three other smaller furnaces in operation. Pictures show lead bullion stacked up in the yards of the larger smelters like cordwood awaiting a favorable lead market.

The two large smelters, whose slag dumps still greet all travelers as they enter Eureka along highway 50, were those of the Richmond Mining Co. at the south end of town and the Eureka Consolidated Mining Co. at the north end. Their mines were adjoining properties 2 miles to the west, on Ruby Hill.

Most of the production occurred in the period 1871–1888, and the Richmond mine and smelter were closed in 1890. The Eureka Consolidated Mining Co. had by this time ceased mining operations on company account but had let out small blocks of ground around the old ore bodies to lessees, or "tributors," as they were called in those days. The Eureka company continued to operate its smelter for several years on the production from lessees but as the production decreased it became unprofitable to continue and the smelter was dismantled.

Of the more than forty-two million dollars produced from the wedge-shaped block of dolomite, over forty million was produced by the Richmond and the Eureka Consolidated Mining companies.

Originally, six companies owned the claims covering the surface exposure of Eldorado dolomite on Ruby Hill but the Richmond absorbed the Albion Consolidated to the west of it and the Eureka Consolidated acquired the K K Consolidated lying to the east, and the four companies then remaining were, from northwest to southeast: Richmond Mining Co., Eureka Consolidated Mining Co., Phoenix Silver Mining Co., Jackson Mining Co.

### *Development Operations*

*Richmond Mining Co.*—After the Richmond Mining Company's first shaft, the Tip Top incline, reached the large, flat thrust-plane ore body, the company started a vertical shaft, the Richmond, at a point on the north side of Ruby Hill, 900 ft to the northeast. This shaft, starting at an elevation more than 150 ft lower than the Tip Top incline, is 1200 ft deep, and the company established levels 100 ft apart until the 900-ft level was reached; below that the levels are 150 ft apart. The shaft started in Secret Canyon shale a short distance to the north of the Ruby Hill fault and on the west side of the Bowman fault; passed through the shale and the Ruby Hill fault between the 2nd and 3rd levels, where it entered the Eldorado dolomite and passed into Prospect Mountain quartzite at 700 ft and bottomed in that formation.

The flat thrust-plane ore body trended downward to the northeast to the 200 ft level of the Richmond mine. There ore was found along the dolomite-quartzite contact and along the Richmond fault as well as on the thrust plane. Below this level, the thrust-plane ore split up, one part trending downward to the northwest while the other part continued downward to the northeast. Through a series of ore stringers and connecting lenses the second part joined with ore along the Richmond fault, passed through the fault into the hanging wall and continued until it reached the top of the Potts Chamber ore body.

It was the latter ore body that caused extended litigation between the Richmond and the Eureka Consolidated companies in the late seventies. The Richmond company contended that it had the right through apex of its ore body at surface to follow that ore body regardless of the fact that in strike the Potts Chamber crossed into the Eureka Consolidated property, and regardless of the fact that the two companies had previously established a boundary line along the end lines of their respective claims. The court held in favor of the Eureka Consolidated, making the compromise boundary line permanent.

*Exploration Beyond the Ruby Hill Fault*—The Richmond company began to probe into the hanging wall of the Ruby Hill fault at its 600-ft level, but as these crosscuts entered formations stratigraphically higher than the Eldorado the work was of very limited extent. However, a crosscut through the fault on the 1050-ft level, at a point west of the Bowman fault, entered a downthrown segment of Eldorado. Several hundred feet of drifting and crosscutting was done in this block but no ore was found. The company then put down five diamond-drill holes from different parts of this development work to prospect the formation at a little greater depth, but all without success. On the 1200-ft level, the company merely crosscut 140 ft into the Eldorado to the northeast of the fault and quit.

*Eureka Consolidated*—The Eureka Consolidated Company's main vertical shaft, the Lawton, was 730 ft deep. It started in Eldorado dolomite and bottomed in Prospect Mountain quartzite. Then at a point 600 ft to the northwest of this shaft, on the 730-ft (9th) level, starting in Eldorado dolomite, the company sank an incline shaft to the 14th level. This shaft also bottomed in the quartzite not far below the Ruby Hill fault. This level was beneath the dolomite wedge and hence found no ore.

The ore bodies in the Eureka Consolidated property also were made along thrust planes in the dolomite, on or near the underlying quartzite. Ore also formed lenses in the dolomite along the contact where the formations were separated by the Richmond fault. Other lenses followed into the dolomite along the Lawton cross fault and eventually connected with the Potts Chamber, as did those of the Richmond farther to the west.

*Exploration in the Hanging Wall of the Fault*—The Eureka Consolidated also ran short crosscuts through the fault into the upper formations. However, with the intention of really exploring beyond the fault for what the company hoped would be the downthrown portions of the surface ore bodies, it sank the vertical Locan shaft to a depth of 1200 ft. It started in Hamburg dolomite 850 ft to the northeast of the Ruby Hill fault and bottomed in Secret Canyon shale.

On the 1200-ft level, the fault separated Secret Canyon shale and Prospect Mountain quartzite. At the point where the crosscut on this level cut the fault, pyrite was present in the fault filling but so much water was encountered that the pumps were drowned and the development ceased.

*Closing Years of Operation*—After the Eureka Consolidated closed its smelter, lessees continued to mine, or "chloride" as they called it, the better streaks and bunches of ore in the iron carbonate casings, which they shipped to the Salt Lake Valley smelters. Sometimes the pay was pretty lean but a good car now and then helped to keep the wolf from the door and a small yearly production was maintained until 1905. In that year the Richmond and the Eureka Consolidated properties were consolidated into the Richmond-Eureka Mining Co. and in this new company the United States Smelting, Refining and Mining Co. held the controlling interest.

In the years between 1905 and 1910, the Richmond-Eureka Mining Co. rehabilitated the Eureka Consolidated and Richmond mines and shipped the stope casings and many of the old stope fills that had formerly been too low in grade. This ore had an average grade of 3 pct lead, 30 pct excess iron over insoluble, with some lime and a gold-silver content of $6.00 per ton. At that time the bulk of the custom ores reaching the Salt Lake Valley smelters were highly siliceous and the basic ores from Eureka supplied a much needed flux.

However, early in 1910, floods washed out the Eureka and Palisade railroad in so many places that several years passed before it was repaired. In the meantime the mine had closed, and when the railroad was again in operation the railroad company had so boosted its rates that the Richmond-Eureka Mining Co. decided it could not operate at a profit on this low-grade product, and the mines remained closed. Lessees continued to chloride a little ore and were able to make a small production after the railroad was operating.

### Renewal of Search

The resumption of exploration for deeper ore bodies dates from 1919, when the Ruby Hill Development Co., financed in Canada, took a lease on the property of the Richmond-Eureka Mining Co. and started to unwater the Locan shaft, with the intention of sinking it further.

At that time, the water stood in the Locan shaft at a depth of about 1000 ft. The company found that it was necessary to retimber the shaft below 600 ft, and it encountered so many difficulties in the pumping operations that when the top of the station was reached at the 1200-ft level its money had given out, and the project was abandoned.

In 1923, the Richmond-Eureka Mining Co. unwatered the Locan shaft to the 1200-ft level. Finding the original crosscut in the Secret Canyon shale very badly caved, it drove a parallel crosscut southwest to the Ruby Hill fault and the adjoining quartzite. It then connected to the old working a short distance to the southeast and also drove a drift to the northwest, partly in the quartzite, until it reached the Geddes limestone (which underlies the Secret Canyon shale). Water stored in the interstices of this thin-bedded limestone drowned the pumps, and this work was stopped. Along the fault, however, mineralized material was found over widths of from 6 in. to 2 ft, and some of it carried valuable deposits of lead and silver with a small amount of gold.

The company then went back to the 841-ft level, above the water; drove a crosscut 200 ft to the northeast of the shaft and there started a vertical diamond-drill hole. This hole started in Hamburg dolomite, passed into Secret Canyon shale at a depth of about 350 ft and entered the Eldorado dolomite somewhere between 700 and 740 ft. However, much caving of the hole had occurred in the bottom part of the Secret Canyon shale and the upper part of the dolomite was badly crushed, so that even after solid dolomite was entered, at 740 ft, the hole caved above the drill. In consequence, when a depth of 760 ft has been reached, the hole was abandoned.

About the year 1926, the Richmond-Eureka Mining Co. sent its geologists, R. N. Hunt and R. T. Walker, to the property to make a detailed geologic study. They made an excellent geological map of the area immediately surrounding the property and recommended that deeper exploration be attempted, but this work never was started.

### Discovery of Ore

In 1935, Thayer Lindsley, in looking over Nevada mining districts, went over the Eureka district hurriedly. He liked the possibilities and later sent geologists to give the area a more thorough study. They, too, liked the chances and in 1937 Dr. W. F. James obtained a long-time lease on the Richmond-Eureka Mining Company's property and an option to purchase the adjoining property on the north, owned by the Cole-Wittenberg interests. This lease and option were then turned over to the Eureka Corporation Limited, which, in the fall of that year, started a vertical diamond-drill hole to explore in advance of sinking.

This drill hole, known as *A* hole, was located far enough in the hanging wall of the Ruby Hill fault to enable it to reach the Prospect Mountain quartzite before penetrating the fault. The geologic plan in Fig 2 shows a cross fault, the Lawton, striking slightly east of north (one of the normal faults), dipping about 70° to the east and ending in the footwall of the Ruby Hill fault. Another fault, the Bowman, with a similar strike and dip, ends at the hanging wall of the Ruby Hill. It was thought that perhaps the Lawton and the Bowman were faulted portions of the same fault, with a horizontal displacement along the Ruby Hill of about 800 ft. Drill hole *A* was so located that if the ore bodies had been shifted that 800 ft (considering the Ruby Hill post-mineral) the drill hole would still have a good chance of penetrating an ore lens.

Diamond-drill hole *A* started in Pogonip limestone 60 ft to the east of the Bowman fault; passed through the fault and into Secret Canyon shale at 200 ft, into Geddes limestone at 832 ft, into Eldorado dolomite at 1190 ft, through the Eldorado into quartz monzonite porphyry at 3202 ft and bottomed still in the porphyry, at 3240 ft. It is thought that the porphyry has been intruded along the contact between the dolomite and the quartzite, but caving of the hole caused abandonment before the quartzite was reached.

It took the E. J. Longyear Co. 14 months to drill this hole, but much experience was gained that was helpful in drilling later holes. No ore was found in the hole but small amounts of lead and zinc occurred in the monzonite porphyry.

Most men would have become discouraged after such a poor showing but Mr. Lindsley, after pausing some months for breath, opened the Locan shaft to the 841-ft level and began to drill again from there.

Diamond-drill hole *B* was started from the 841-ft level of the Locan shaft, at a point 430 ft to the northeast. It was thought at the time that it would be farther from this level to the top of the Eldorado dolomite than it had been from the surface to the top of the formation in *A* hole, because in *B* the hole was starting in Hamburg dolomite while in *A* hole Secret Canyon shale, which was cut at 200 ft, extended to the surface on the west side of the Bowman fault. Furthermore, erosion had removed the top of it opposite *A* hole. However, in drilling *B* hole it was found that all the formations, at least those below the Hamburg, had been considerably thinned by the thrusting. The Secret Canyon and Eldorado were found to be less than half their normal thickness and the Geddes limestone was thinned to a thickness of only 10 ft at this point. This hole penetrated the Prospect Mountain quartzite at 1690 ft after passing through about 14 ft of quartz monzonite porphyry, and continued in the quartzite to a depth of 1900 ft.

*B* hole passed through a 15-ft section

of mostly oxidized iron-zinc ore beginning at 1152 ft, and at 1409 ft it entered a 40-ft width of solid sulphides of iron, zinc and lead. Below that, in the dolomite, it passed through several apparently vertical streaks of sulphide ore before reaching the porphyry.

After *B* hole had been finished, the drill hole abandoned by the Richmond-Eureka Mining Co. in 1924 was reopened and drilled from 760 ft to a depth of 1562 ft. It is called *C* hole, and passed from Eldorado dolomite into monzonite porphyry at 1522 and into quartzite at 1554 ft. This hole encountered sulphides in the dolomite at a depth of 1340 ft, with occasional streaks of sulphide ½ in. thick, and at 1360 ft it entered solid sulphide ore, similar to that in *B* hole, which continued for 30 ft.

In November 1941, preparations were begun for sinking a new shaft, the Fad. Sinking was started in February of the following year. It is 1430 ft to the northeast of the Locan shaft and at an elevation 50 ft lower. Sinking of the shaft went forward until water was encountered at 235 ft in the Pogonip limestone, then, although the flow never was more than 60 gal per minute, war had made it impossible to obtain proper pumps and by the time the shaft had reached a depth of 540 ft its progress had slowed to almost nothing.

### *Ore Body Confirmed*

In the fall of 1943, the Bureau of Mines undertook to drill additional holes from the 841-ft level of the Locan shaft, to prove whether or not the ore found in drill holes *B* and *C* was a real ore-body or whether the holes had accidentally cut two isolated spots of ore. Holes *D*, *E* and *F* were drilled, respectively 200 ft north, 200 ft east and 200 ft west of *B* hole. Hole *F* is also 200 ft north of *C* hole and the pattern of the five holes is that of a diamond. All these holes penetrated what is apparently the same sulphide ore body as that found in drill holes *B* and *C*, and together they proved almost conclusively that a real ore body had been discovered.

The results of assays from sludges from the five drill holes show average assay values as follows:

*Drill Hole B*—Across 40 ft from 1409 to 1449 ft, assayed gold 0.164 oz, silver 3.56 oz, lead 2.46 pct, zinc 14.44 pct. Another zone in this hole, of 15 ft from 1494 to 1509 ft, assayed gold 0.123 oz, silver 2.35 oz, lead 2.90 pct and zinc 4.74 pct.

*Drill Hole C*—Across 40 ft from 1350 to 1390 ft, assayed gold 0.139 oz, silver 6.75 oz, lead 4.51 pct and zinc 10.05 pct. Another zone in this hole of 18 ft (from 1425 to 1443 ft) assayed gold 0.104 oz, silver 6.29 oz, lead 3.14 pct and zinc 3.82 pct.

*Drill Hole D*—Across 27½ ft from 1469 to 1496.5 ft, assayed gold 0.11 oz, silver 3.29 oz, lead 2.33 pct and zinc 8.33 pct. Contiguous to this 27½ ft is a width of 25½ ft of lower grade ore.

*Drill Hole E*—Across 59½ ft from 1502½ to 1562 ft, assayed gold 0.42 oz, silver 12.12 oz, lead 8.23 pct and zinc 11.85 pct.

*Drill Hole F*—Across 27.4 ft from 1375.6 to 1403 ft, assayed gold 0.153 oz, silver 3.49 oz, lead 2.22 pct and zinc 6.35 pct.

### PRESENT OPERATIONS

In none of the drill holes was it possible to obtain core samples of the ore except specimens attached to cement, but the amount of sludge obtained was nearly the amount of a calculated hole.

Pyrite, some arsenopyrite, and some dolomite were the gangue minerals, and lead, zinc and pyrite concentrates will be made by selective flotation. The pyrite, which contains most of the gold, will then be roasted and cyanided.

During the year 1945, a surface plant with equipment capable of sinking the Fad shaft to the depth of the sulphide ore body was installed and unwatering of the shaft was started.

Originally, the Fad was timbered with 8 by 8 wooden sets. The three south compartments were 5 by 5½ ft in the clear and the north or manway compartment was 4½ by 5½ ft in the clear. This made the overall dimensions of the shaft, including the 2-in. lagging, 23 ft 2 in. long by 7 ft 2 in. wide.

When work was resumed in 1945, it was found that the wooden sets below the water level, which at that time was about 300 ft below the surface, had settled out of line; so, starting just above the water, the wooden sets were removed and most of the section below that point was concreted. At that time it was nearly impossible to obtain timber for sets, so it was decided to use steel sets; although the cost is greater, the probable life is also greater. These sets, starting just above 300 ft, are made from 6-in. I-beams placed at 6-ft centers. In order to conform with the 8 by 8 wooden sets above, the three south compartments are 5 ft 2 in. by 5½ ft in the clear and the manway is 4 ft 8 in. by 5½ ft in the clear. The overall dimensions are 22 ft 8 in. by 6½ ft, and, since the lagging is wedged inside the 6-in. I-beams, no extra shaft space is required.

The shaft passed from Pogonip limestone into Dunderberg shale at 800 ft (because of shale within the Pogonip, I had placed this contact at a higher elevation). It passed from Dunderberg shale into Hamburg dolomite at 1080 ft and is still in the Hamburg at the present depth of 1585 ft.

The water in the shaft increased gradually from 60 gal per min. at 500 ft to 600 gal at 1200 ft and to nearly 1000 gal at 1585 ft. However, measurements in the diamond-drill holes show that the area has drained at a rapid rate.

At 790 ft a connection was made to the 841-ft level of the Locan shaft and a main pumping station was constructed at this level. At 1233 ft a temporary pump station was cut and at 1680 ft another main pump station will be cut. At 2250 ft a main pump station will be cut and a crosscut will be driven to the ore.

## Acknowledgments

During the summer of 1932, Dr. T. B. Nolan, of the U.S. Geological Survey, started a survey of the Eureka district. Lack of funds held up the work until 1938, when Dr. Nolan returned. He then spent the summers of 1938, 1939, 1940, 1941 and 1946 in the district and has now completed his field work. Assisting him on palentology were Dr. Josiah Bridge, Dr. C. W. Merriam, Dr. Fisher, Dr. Cooper and Dr. Cloud at various times, and to Dr. Nolan and all of the others the author is indebted for great help.

He is indebted also to Dr. H. E. Wheeler, Dr. Dwight M. Lemmon, Mr. R. N. Hunt, Mr. R. T. Walker, Mr. E. O. Binyon, the Bureau of Mines, and many others.

His thanks for permission to publish this report are given to Mr. George W. Tower, president of Eureka Corporation Limited, and to Mr. George W. Mitchell, general manager of the company.

# Occurrence of Lead-zinc Ore at Iron King Mine, Prescott, Arizona

By H. F. Mills,* Member AIME

The Iron King mine is 12 miles east of Prescott Arizona, in low foothills of the Agua Fria mining district. It was operated in 1906 and 1907, using gravity methods of concentration, mostly on oxide ores. In 1923 and 1924 shipments of sulphide ores were made to the Humboldt smelter. It remained inactive and stripped until 1936, when it was unwatered and sampled to the 400-ft level, the bottom at that time.

In 1937 the Iron King Mining Co. purchased the mine and shipped oxide ores. During 1938 a bulk flotation plant of 140 tons daily capacity was erected. The mill has been enlarged and operations have been continuous since the mill first started.

In July 1942, the Shattuck Denn Mining Corporation purchased the physical assets from the Iron King Mining Co., and since that time it has operated as the Iron King Branch of the Shattuck Denn Mining Corporation. Economic metals recovered are gold, zinc, silver, lead, a little copper, and fluxing pyrite.

The ore occurs as a series of northward-dipping shoots arranged in echelon pattern, the series extending over a length of 3500 ft. Strike lengths of shoots vary from 60 to 600 ft, and widths from 3 ft to as much as 14 ft. The ore occurs in a sheared zone in pre-Cambrian chloritic schist. This schist belt is about 6 miles wide, and the ore zone is about 4 miles from the pre-Cambrian granites to the west. No intrusive rocks have been found in the mine, nor is there evidence of any on surface that might indicate a genetic connection with the ore occurrence (Fig 1).

The schists strike N23°E and dip 75° to the northwest, which might indicate an overturned thrusting from the granites. In the upper levels, the echelon pattern in which the ore occurs was quite regular, no disturbance of the enclosing schist, or of any faulting that might contribute to the formation of the ore pattern was noted. There is a possibility that the ore zone may be on the axis of an isoclinal fold.

Another interesting feature is the more or less typical zoning of minerals along the strike in each of the ore shoots (Fig 2). The north end of each shoot is enclosed by schist, while the south ends usually extend out at full width beyond the economic mining limits in a pyrite-quartz low-grade material.

The plane of demarcation between ore and schist is very sharp, with little or no impregnation of sulphide in the schist, especially on the footwall side. The footwall schists are usually firm and strong, and vary from dioritic phases to fine-grained, lighter colored bands, almost silky in texture. The hanging wall for a width of 50 to 80 ft consists of alternating sheetings of chloritic and pyritic schists.

The mine is now partly developed to the 920-ft level, with some diamond drilling from the 1150-ft station of No. 6 shaft. Economically there has been little change in nature or grade of ore from the top of the sulphides, about 200 ft below surface, to the bottom levels. There have been a few echelons in depth, just as there are in plan, and while a few shoots have had

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Presented at the Arizona Section Meeting, Tucson, Oct. 28–30, 1946. Manuscript received at the office of the Institute Nov. 12, 1947. Issued as TP 2190 in Mining Technology, July 1947.

* Manager, Iron King Branch, Shattuck Denn Mining Corporation, Humboldt, Arizona.

marked changes with depth, the pattern as a whole has somewhat increased in area and available tonnage with depth. Because of premium metal payment, some ores have been mined that would not have been extracted on ceiling prices, accounting for an increase in tonnage from the lower levels, but this was done with no appreciable lowering of assay content per ton of ore.

From the structural aspect, however, a number of changes have been noted with depth (Fig 3). It is evident that the echelon pattern has been formed by a series of nearly strike and dip faulting. The shear planes or zones bear slightly more east of north than the ore, and usually dip slightly flatter than the ore shoot. In the upper levels, and more particularly on the south end of the mine, the shearing may have been of lesser degree and have been recemented. When an overlap was formed under these conditions, it was rare that there was more than one strand of commercial grade of ore at a given point along the strike. At depth, the shear zones are more discernible, and probably of greater movement, resulting in as high as three minable strands of ore due to overlapping. It is quite certain, too, that the whole zone, at least for a width of 100 ft, has been sheared, the general movement having been taken up a little in each of thousands of schist partings in addition to the major shears that have produced the echelon pattern.

The intersections of the fault shears and the vein plunge northward, except in one instance in which the fault is steeper than the vein and produces a south rake of the faulted segment.

The zoning of minerals in each shoot is of interest, not only because it presented some difficulties in trying to maintain a uniform mill feed, but also because it might have a bearing on the genesis of the ore. E. H. Crabtree did some flotation

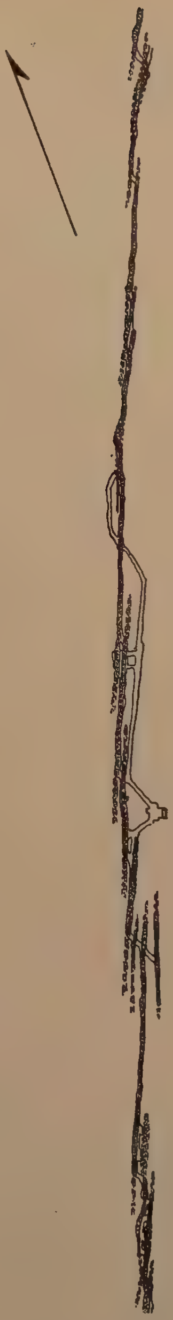

Fig 1—Ore series.

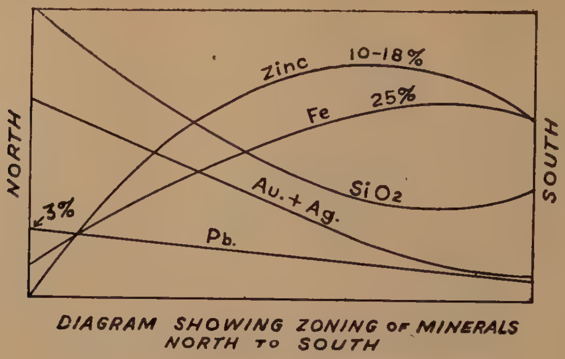

FIG 2—ZONING OF MINERALS NORTH TO SOUTH.

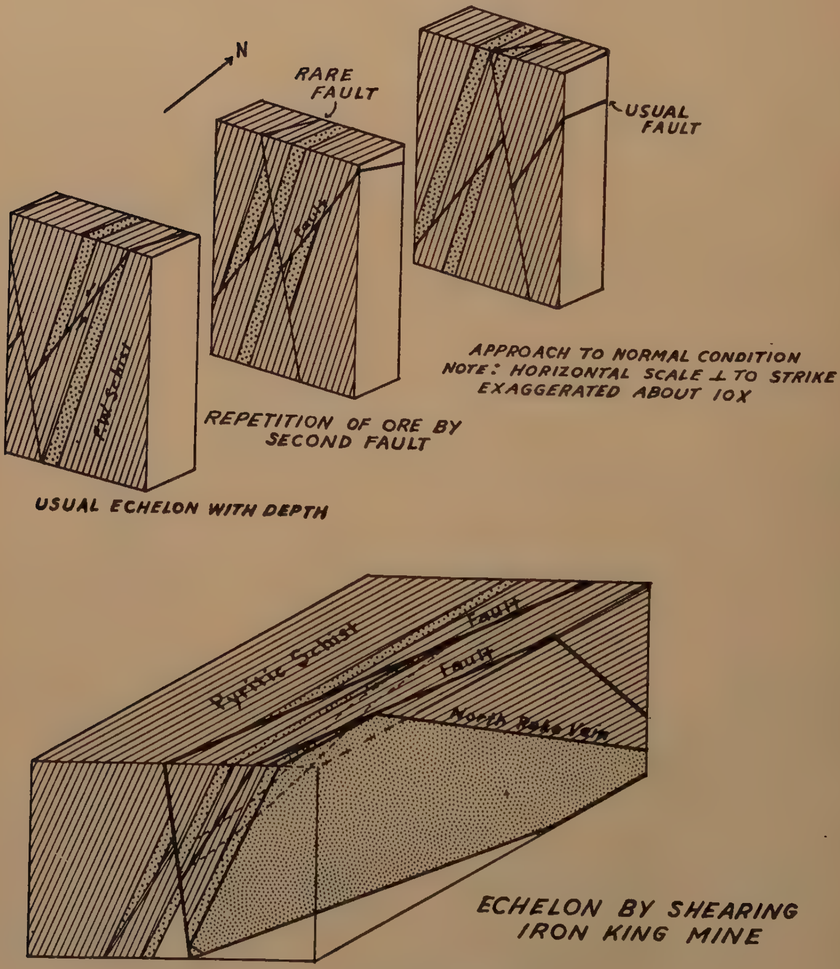
N
RARE FAULT
USUAL FAULT
Fault
F.W. Schist
APPROACH TO NORMAL CONDITION
NOTE: HORIZONTAL SCALE ⊥ TO STRIKE EXAGGERATED ABOUT 10X
REPETITION OF ORE BY SECOND FAULT
USUAL ECHELON WITH DEPTH
Pyritic Schist
Fault
Fault
North Rake Vein
ECHELON BY SHEARING IRON KING MINE

FIG 3—CHANGES WITH DEPTH.

tests and microscopic work on the ore in 1939 and observed the following order of deposition: pyrite, arsenopyrite, chalcopyrite and sphalerite, tennantite, galena and gold, and gangue.

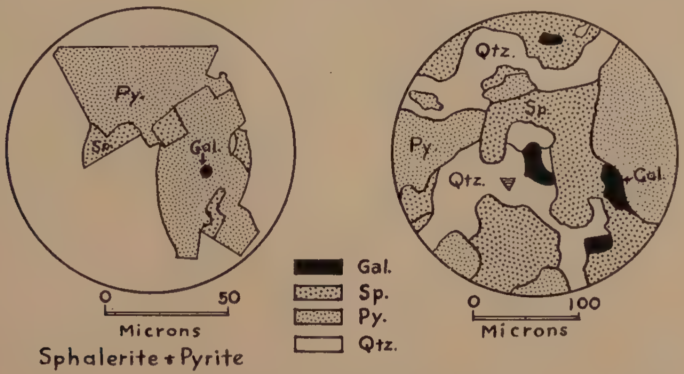

FIG 4—DISTRIBUTION OF MINERALS.

The ore is usually hard, dense and abrasive. As shown in Fig 4, the minerals are finely crystallized, with considerable intergrowth of sphalerite into pyrite and interlocking to a lesser extent between galena, sphalerite, gangue and pyrite. Microscopic gold is seen occasionally with galena or sphalerite. Comparison of specimens from the same ore shoots but at various depths has shown no appreciable increase in crystal size or decrease in interlock at depth.

Pyrite constitutes nearly half of the average mine ore and carries a relatively large part of the gold. Silver has a definite relation to copper and is present to some extent at least as tennantite. The zinc varies in range from light-colored, free floating to dark-colored marmatites. Chalcopyrite and arsenopyrite are present in small quantities and also constitute good host minerals for gold and silver.

The footwall at the north end of each shoot becomes softer than usual with some bleaching and sericitization. We consider now that the original ore deposition was a pyritic replacement of either an intensely sheared narrow zone or of a certain strata of schist amenable to replacement. This was followed by a certain amount of the present displacement with introduction of sphalerite. This mineralization was controlled to some extent by the faults, with replacement of pyrite by sphalerite. Later this zinc was replaced by a gold-silver and quartz type of mineralization. The footwall at the north ends of the ore shoots is altered then, not only from proximity to the echelon-producing faults, but also because it has been subjected to all phases of mineralization.

It is possible that no magmatic source of mineralization will be found within future mining limits at depth, although the formation has been subject to more distortion and fracturing on the 900-ft level than on any of the levels above it.

The mine has made about 30 gal of water a minute during its entire operation. This water comes from the bottom, is cool and has an alkalinity slightly above neutral.

## DISCUSSION

HUGH M. ROBERTS*—Seldom does a mine manager show any real disposition to arrive at an understanding of the underlying geologic relations that have to do with his ore bodies.

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* Mining Geologist, Duluth, Minn.

Too often these questions are shoved off to the limbo of the specialist. But Mr. Mills has been inquiring enough to want to know why his ore lenses are arranged en echelon, why the metals are distributed as they are with respect to terminations of the ore shoots and to know just why the lenses of ore are doubled and trebled on some levels. So much the better then, has he been able to plan his stoping operations and meet the milling problems that have arisen at the Iron King Mine.

Reading his paper, one has a feeling that because of his keen insight into the geology of the mine, no ore will be missed and that it will be mined advantageously. To be effective, any mining operation must be directed in accord with the nature of the ore bodies, the variety of minerals contained in the ore, the probable extent thereof in depth, the configuration of the ore shoots, that is, in conformity with all phases of the geology of the ore occurrence.

Mr. Mills has thought his way through some difficult geological problems to the advantage of his overall operation. Also he has noted his findings in brief, readable compass, which will be of material aid to the next miner or geologist who is called upon to deal with an analogous situation.

# Stratigraphy of the Mascot-Jefferson City Zinc District

By Charles R. L. Oder,* Member, and Howard W. Miller,† Junior Member A.I.M.E.

## Abstract

Approximately 5000 tons of zinc ore a day was mined during 1943 in the Mascot-Jefferson City district in East Tennessee. This ore came from the Kingsport formation, a part of the Knox dolomite, of Ordovician age. The Kingsport is composed of 350 to 400 ft. of limestone and dolomite. Thin cherts, sandy layers and shales are numerous. Frequently, the limestones are altered to crystalline dolomite. The Kingsport is divisible into four main lithologic zones. These divisions are illustrated in an accompanying table, which gives detailed sections of the ore-bearing beds as found in the mines and shows the exact positions of the various ore zones within the formation. Key beds are described and the practical applications of stratigraphy in mapping, prospecting and mining are discussed.

## Introduction

The Mascot-Jefferson City zinc district is in Knox and Jefferson Counties, in the Great Valley of East Tennessee. It extends from Mascot, 13 miles northeast of Knoxville, to Jefferson City, about 30 miles northeast of that city.

The American Zinc Company of Tennessee is producing from five mines in the district, in which it has operated for more than 30 years. The only other important producer is the Universal Exploration Co. The approximate daily production of the district in 1943 was 5000 tons of sulphide ore, grading from 2.5 to 6 per cent zinc. Small amounts of oxidized ore are mined intermittently.

The Knox dolomite has been described by various geologists,[1–4] but no detailed descriptions of the rock sections revealed by mining have been published. Rock units are not conspicuously different and careful study is required to distinguish them. Nevertheless, familiarity with the stratigraphy of the mineralized zones is essential to successful exploration and extraction of the ores; therefore the purpose of this paper is to give detailed stratigraphic sections as measured in the mines, to discuss their important features and to show their relationship to the stratigraphy of the district.

## Generalized Section

Table 1 gives the most characteristic general features of the major stratigraphic units. The formation names given in parenthesis are locally and widely used terms, the correlations of which are not necessarily established. For this reason they are put in quotation marks and are replaced by more restricted terms. The names "Mascot" and "Kingsport" were adopted from local towns. They were chosen jointly by Josiah Bridge, of the Federal Geological Survey, and the senior author for use in their separate unpublished manuscripts describing the Knox dolomite in East Tennessee. Descriptions of the type sections may soon be published. The name "Longview" was first applied by Butts[5] in Alabama.

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Manuscript received at the office of the Institute Sept. 20, 1944. Issued as TP 1818 in Mining Technology, May 1945.

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† Geologist, Grasselli Mine, American Zinc Company of Tennessee, New Market, Tennessee.

[1] References are at the end of the paper.

Zinc sulphide mineralization occurs in various parts of the Knox dolomite in East Tennessee, but the only known commercial ore in this district is in the Kingsport formation; therefore the generalized section is meant to show the stratigraphic position of that unit and to compare its general character with that of overlying and underlying formations.

TABLE 1.—*Generalized Section, Mascot-Jefferson City Zinc District, East Tennessee*

| Age | | | Formation Name | Thickness, Ft. | Character of Rocks |
|---|---|---|---|---|---|
| Ordovician | Chazyan | | Lenoir and Mosheim ls. (Chickamauga) | 250± | Dark gray (Lenoir) and dove colored bird's-eye (Mosheim) limestone |
| | Canadian | Knox Dolomite | Mascot ("Cotter-Powell") | 500–700 | Light and dark gray dolomite and limestone. Moderately cherty. Limestone most abundant in Jefferson City area. Base marked by chert matrix sandstone. Rarely mineralized |
| | | | Kingsport ("Jefferson City") | 350–400 | Dark to very light gray fine-grained dolomite and dove to brown limestone. Upper 200 ft. mostly dolomite. Chert and arenaceous zones numerous. Limestone locally altered to crystalline dolomite. Main ore horizons in lower two thirds of formation |
| | | | Longview ("Nittany") | 250± | Highly cherty, light gray dolomite |
| Cambrian | Ozarkian | | Chapultepec | 500± | Mostly dolomite at Mascot. Light and dark gray dolomite and limestone in Jefferson City area. Very cherty. Heavy sandstone at base |
| | | | Conococheague Copper Ridge | 900–1200 | Dark sugary and light gray fine-grained cherty dolomite. Some limestone in Jefferson City area |
| | Croxian | | Maynardville ls. | 60–250 | Light blue to dark gray laminated limestone and dolomite |
| | | | Nolichucky shale | 550–750 | Dark gray, greenish and yellow shales and thin limestones |

## GENERAL FEATURES OF THE KINGSPORT FORMATION

The Kingsport is made up of 350 to 400 ft. of limestone and dolomite. The limestones are dove colored to brownish gray and are fine grained. The dolomites range in color from light to dark gray and in texture from fine grained to coarsely crystalline.

Arenaceous layers are numerous. They range in thickness from one inch to one foot, and from little more than a few scattered sand grains in the dolomite or limestone to a true sandstone. The quartz grains are frosted and well rounded. Greenish gray shale seams also are numerous but seldom more than an inch thick. These shales usually divide dolomite or limestone beds of slightly different physical characteristics.

The Kingsport is moderately fossiliferous. Casts of gastropods, opercula, cephalopod siphuncles, and brachiopods make up the principal forms. The most accessible specimens are species of the operculum, *Ceratopea*, and the "pipe stem" siphuncles, which may be found free in the soil. Fossil fragments rarely are seen in mine sections and diamond-drill core.

Most of the Kingsport chert occurs as white to pearl gray, concentrically banded nodules, which vary from 1 to 4 in. in thickness and are elongated parallel to the bedding. Much of this nodular chert is believed to be syngenetic in origin because: (1) it is confined to certain beds in the numerous localities where it has been observed, (2) it is involved in faulting and brecciation in such a manner as to indicate that it antedated local structural deformation, and (3) it displays no characteristics that suggest a secondary genesis. In addition to the nodular "flints," there is considerable white, brown and bluish gray chert, which forms beds and irregular masses one inch to more than 2 ft. thick, especially in the lower part of the formation. The following features of this material and circumstances surrounding it are considered evidence of its representing replacement of limestone or dolomite by silica: (1) it is not confined to any certain strata; it is discontinuous and shows up often in beds ordinarily void of chert, (2) it is most abundant in the vicinities of fracturing and other forms of structural deformation, and (3) it contains casts of dolomite rhombs, preservation of banding and laminae which extend into and coalesce with similar structures in unreplaced dolomite, and inclusions or remnants of dolomite that are very similar to the strata in which it is found.

### *Main Divisions*

On the basis of lithology, the Kingsport formation may be divided into four main divisions (see Table 2). The upper division is composed of 101 to 135 ft. of fairly light to dark fine-grained dolomite, with a little interbedded brown limestone. Arenaceous beds and dark nodular cherts are numerous.

The underlying 44 to 50 ft. consists of light brownish gray to nearly white, extremely fine-grained dolomite, some of which is nearly lithographic in texture. Gray concentrically banded chert nodules are characteristic.

The light colored division just described is underlain by a variable zone 33 to 38 ft. thick. It is made up of light to dark gray to brownish fine-grained to finely crystalline dolomite and some brown limestone, which may be altered to crystalline dolomite.

The lower 178 to 216 ft. of the formation forms the fourth division. It consists mainly of brown limestone, which in many places is altered to crystalline dolomite. Interbedded with the limestone is some light to dark gray fine-grained dolomite. Cherty layers, especially of the irregular and banded type, are numerous.

### *Dolomitization and Crystalline Textures*

The Kingsport formation is composed of three principal rock types: fine-grained limestone, fine-grained dolomite, and coarsely crystalline dolomite, the texture of which varies.

The fine-grained dolomites are more or less constant in character in both normal undisturbed areas and in localities affected by structural disturbance and metamorphism. They occur in nearly all of the Knox belts in East Tennessee and they persist for great distances along the strike. It is believed that they, as dolomites, antedate the tectonics and the ore deposition.

The fine-grained limestones are frequently altered to crystalline dolomites, the textures of which vary from fine to coarse. Locally, the term "recrystalline" has been applied to these crystalline dolomites. The more impressive "recrystalline," the variety that inspired the name, is coarse and marble-like and approaches theoretical dolomite in composition. It is a prominent ore-bearing rock. Usually it is massive and varies in color from light to dark gray. Well-crystallized white gangue dolomite is generally present as disseminations, irregular areas and fracture fillings. In places, it constitutes 50 per cent or more of the rock. Dark, shaly partings are commonly scattered irregularly through the rock.

Much of the "recrystalline" is fine-to-medium grained, although of a distinctly crystalline texture and dolomitic composition. It is either uniformly brown or varies from medium to fairly dark gray and shows a minimum of white crystalline gangue dolomite and dark partings. In many places the differences in the varieties of "recrystalline" are hard to distinguish, because of gradations in texture and color.

We believe that the more coarsely crystalline dolomites resulted from replacement of the limestones by magnesian solutions, which probably circulated along open bedding planes and faulted, fractured and brecciated areas. Mining has exposed many places in which the "recrystalline" may be followed, along the same stratum, from the completely altered, well-crystallized and dolomitized rock into totally unaltered fine-grained limestone. Such characteristics as banding and shaly partings often can be followed from unaltered limestone into medium to coarsely crystalline dolomite.

## DETAILED SECTIONS AND CORRELATIONS

### *Construction of Table 2*

Rock sections exposed by mining at Mascot, New Market* and Jefferson City are graphically described in Table 2. The measurements were made in great detail. Individual layers often were treated as distinct units. The bed that first was considered the footwall of the known ore in each mine was assumed as the datum or zero bed. Each unit above or below that bed was given a number corresponding to the actual measured interval between the datum and the unit being described. For example: an important black dolomite, 28 ft. above the zero bed at New Market, is called the "28 bed." Experience has shown that these numbers form names that are readily grasped and that automatically designate intervals.

The correlation of the mine sections with surface geologic mapping and surface drill-hole data has established the stratigraphic positions of the important ore zones and the mine sections in the Kingsport formation.

### *The Mascot Section*

The greater part of the ore mined at Mascot occurs immediately above a light gray, fine-grained dolomite, which is 3 ft. thick, and which at the base carries 2 in. of scattered quartz grains. Several years ago this became known as the "footwall sand," or "zero bed."

Another widely used horizon occurs from 27 to 33 ft. above the zero bed. It is fine-grained dolomite of which the basal 2 ft. are dark, the middle 2 ft. light and the upper 2 ft. medium gray. Small white spots of crystalline dolomite are scattered through the lower 4 ft. and the upper 2 ft. are mottled by irregular, wavy, dark lines.

An important zone lies 82 to 87 ft. above the zero bed. It is here called the "Grasselli dolomite" because of its prominent position in the ore section in the Grasselli mine at New Market. This bed is composed of dark, fine-grained to finely crystalline dolomite, through which are scattered a few white spots and at the base of which occur green shale and some quartz-sand grains. It is further recognizable because it is underlain by brown limestone, which may be altered to crystalline dolomite, and because it is overlain by 3 to 5 ft. of fine-grained to finely crystalline, mottled and sparsely spotted, greenish to pinkish medium gray dolomite.

The 100–105 bed is an outstanding light brown, fine-grained dolomite characterized by numerous scattered white spots of crystalline dolomite. The top and bottom are generally marked by thin green shale. Small spots of dark gray chert are sometimes found about one foot above the base.

The 120 sand is a very thin but persistent layer of light brown, finely banded dolomite, which is nearly lithographic in texture and through which are disseminated rounded and frosted sand grains.

The character of the 165 bed is indicated by the fact that it is known as the "minus C oölite" in surface prospect drilling. It is a band of blue to brownish, oölitic chert, 2 to 6 in. thick. The oölites vary greatly in size and shape and range in color from white to pearl gray or brown. Usually they have well-developed rings. This bed is characteristically overlain by medium gray to dark gray dolomite and underlain by very light gray dolomite.

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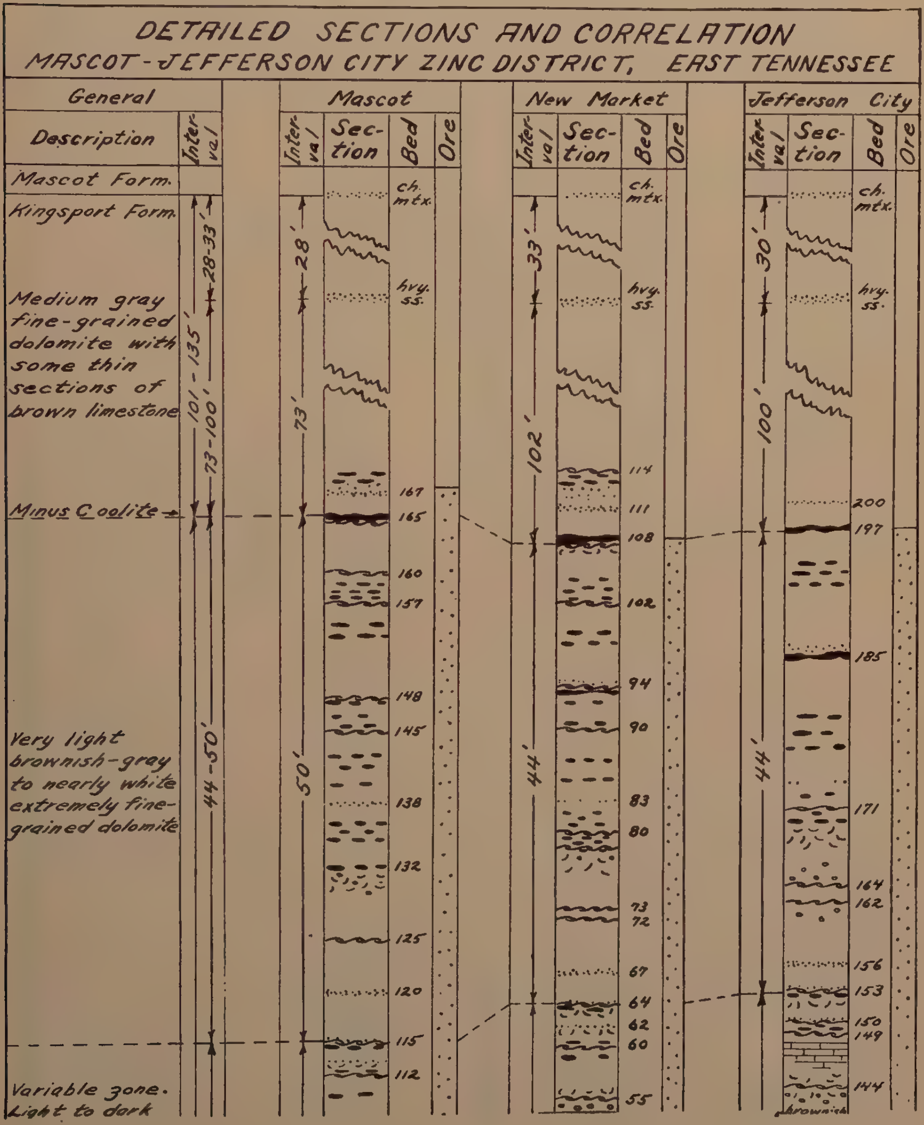
* New Market is about 14 miles northeast of Mascot.

The "heavy sand" varies in position from 25 to 35 ft. below the top of the Kingsport. It is a layer of rounded and frosted, uniformly medium sized quartz grains, 2 to 6 in. thick. This sand is relatively pure, unlike most of the other Kingsport sands in which the grains are disseminated in the dolomite. Float blocks of this bed are common surface residuals.

The Kingsport is separated from the overlying Mascot formation by the chert matrix sand. This sand usually is about

TABLE 2.—*Detailed Sections and Correlation, Mascot-Jefferson City Zinc District*



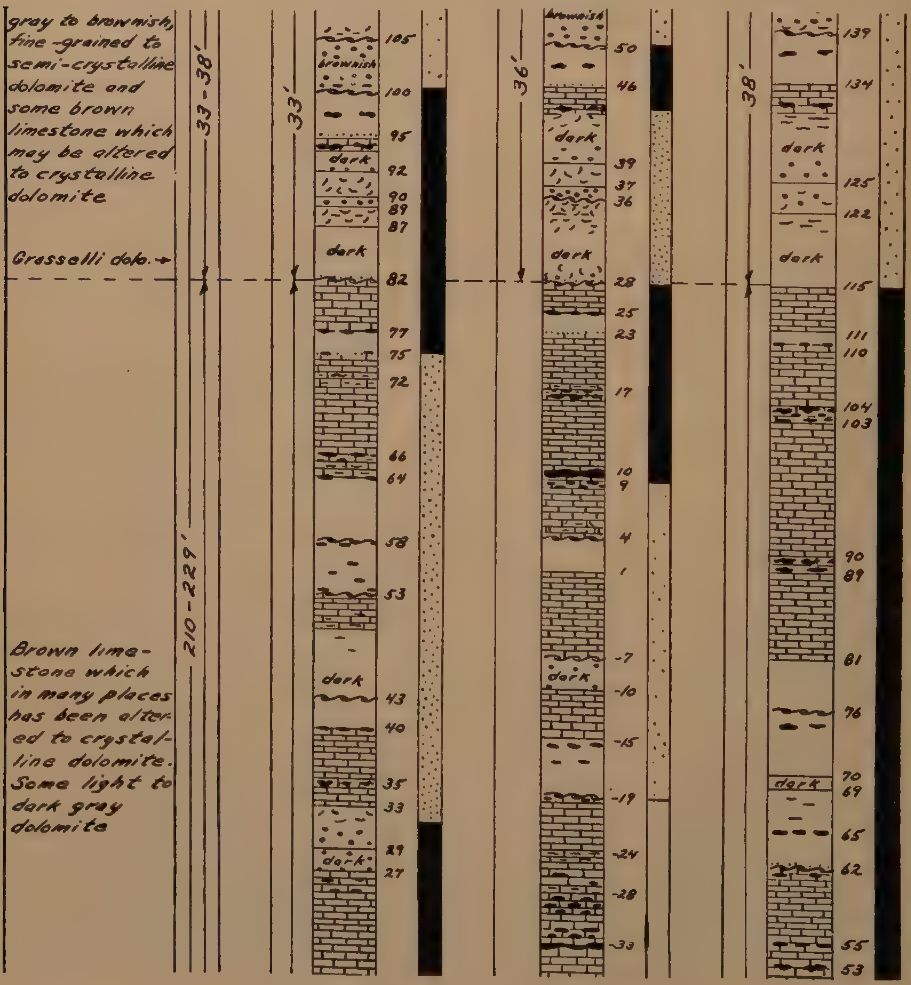
6 in. thick, composed of rounded and frosted, uniformly medium sized quartz grains. Its distinctive feature is an irregular band of snow white chert, forming a cement for part of the sand grains. Surface residuals of this bed are common.

### *New Market Section*

The following key beds are useful and easily recognized at New Market: the chert-matrix sandstone, the heavy sandstone, the minus C oölite (108 bed), the 67 sand, the 50–55 bed, and the Grasselli dolomite (28–36 bed). As shown in Table 2, they can be correlated with similar zones described in the Mascot section.

The most striking and useful zonal marker below the Grasselli dolomite is the thickness from the 9 to the 10 bed. It is usually called the 9–10 bed. The 10 bed is a 2 to 4-in. band of blue chert spotted with white dolomite. Eight to 10 in. below it is a double band of pearl-gray concentrically banded chert nodules. The enclosing rock generally is a brownish, somewhat banded, finely crystalline dolo-

TABLE 2.—*Detailed Sections and Correlation,*


gray to brownish, fine-grained to semi-crystalline dolomite and some brown limestone which may be altered to crystalline dolomite
Grasselli dolo.
Brown limestone which in many places has been altered to crystalline dolomite. Some light to dark gray dolomite
33-38'
210-229'
33'
105
brownish
100
95
dark
92
90
89
87
dark
82
77
75
72
66
64
58
53
dark
43
40
35
33
29
dark
27
36'
brownish
50
46
dark
39
37
36
dark
28
25
23
17
10
9
4
1
-7
dark
-10
-15
-19
-24
-28
-33
38'
139
134
dark
125
122
dark
115
111
110
104
103
90
89
81
76
dark
70
69
65
62
55
53

ite.m Below the 9–10 bed, the section is rather uniformly brown limestone with occasional interbedded dolomite. The latter is finely crystalline in the few mine exposures but evidence found in surface prospect holes indicates that actually it is an alteration of brown limestone. Some holes show more than 125 ft. of continuous limestone below the Grasselli dolomite. Nonpersistent, secondary chert forms a good part of the chert in these beds. It is very strongly developed locally below the minus-19 bed.

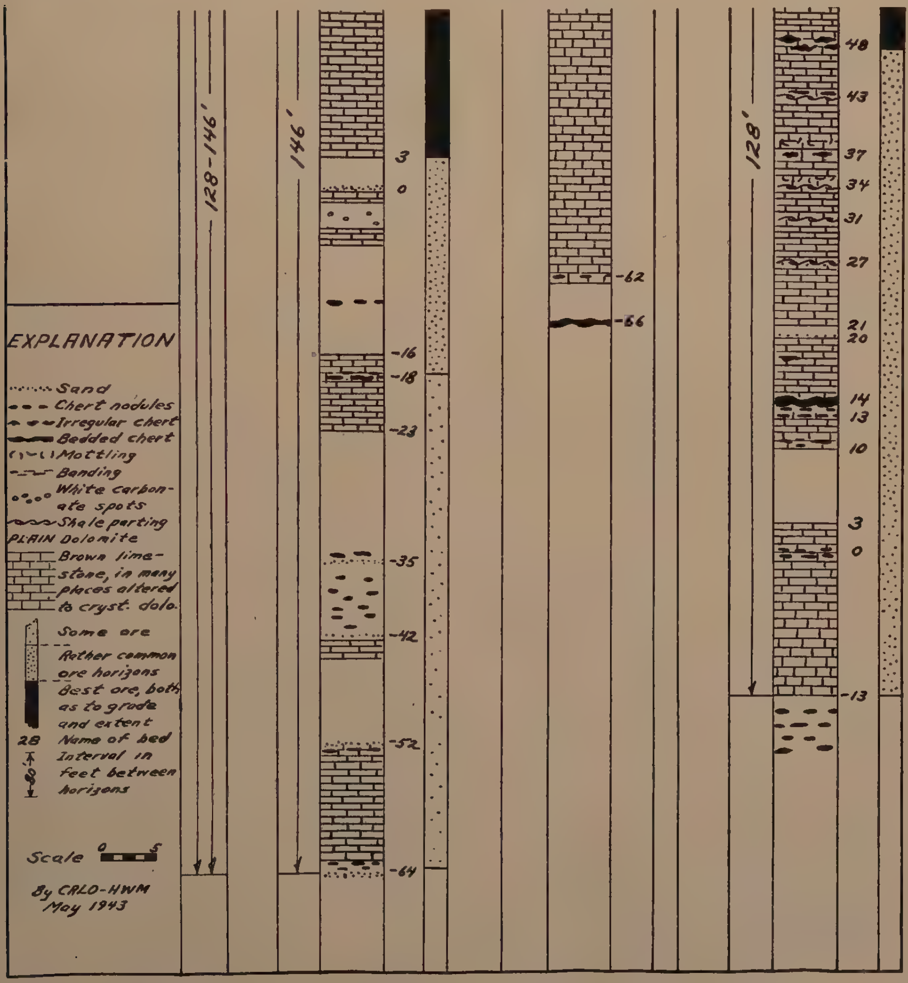
*Jefferson City Section*

The key beds lying above the Grasselli dolomite at Jefferson City are very similar to the corresponding beds at Mascot and New Market, and just as striking.

Below the Grasselli dolomite (115 bed), the section is predominantly limestone. It is believed that even the 62–81 beds, shown in Table 2 as dolomite, represent dolomitized limestone.

The 103–104 bed makes one of the useful zonal markers below the Grasselli dolomite.

*Mascot-Jefferson City Zinc District.—(Continued)*



Generally it is a double row of white to gray, irregular chert nodules in brownish, finely banded dolomite. Lower in the section, the 69–70 bed, a dark, finely crystalline dolomite, is often useful. One of the outstanding of the lower zones is formed by the 3 to minus-13 limestone. It is easily recognized because the beds just above it and below it are steel to medium gray, fine-grained dolomite. The underlying dolomite is characterized by abundant gray to white chert nodules.

### *Correlations*

In the three mine sections, detailed correlation of important zones above the Grasselli dolomite has been made with confidence. This is verified by the records of Table 2. Even very small units, such as the thinly scattered Mascot 120 sand, have the same appearance and relations at Mascot as at Jefferson City, 18 miles away.

The cherts immediately beneath the Grasselli dolomite constitute practically the only zones that can be correlated with ease and confidence below that bed. This lower part of the section is characterized by the predominance of limestone in each of the areas. General similarities exist in this zone but as a rule bed by bed correlations cannot be made among the various limestone, chert and dolomite layers.

## ORE ZONES

The ore in the Mascot-Jefferson City district is simple mineralogically. Sphalerite is the only ore mineral. The gangue is composed principally of white crystalline dolomite accompanied by rare occurrences of small quantities of white calcite. The sphalerite and gangue appear either as material cementing breccia and filling fractures in the dolomite rock or as replacement of the crystalline dolomitized limestone. Generally the replacement ore is richer than that in the breccia.

A glance at Table 2 shows that the stratigraphic extent of the ore ranges from a point about 100 ft. to one about 330 ft. below the top of the Kingsport formation. Only occasionally does as much as 150 ft. of this thickness bear ore at any one place. The commoner ore thicknesses vary from 8 to 40 ft. As shown in Table 2, certain parts of the formation in each area are more favorable to ore occurrence than others. Local structural conditions appear to be primarily responsible for the degree of mineralization in the various zones. These two features, stratigraphy and structure, are inseparably related in all the ore deposits of the district.

### *Mascot*

At Mascot, the most prolific ore zone occurs between the 3 and 29 beds. The section between the 75 and 100 beds also is a good producer. At any one place either or both zones may contain ore and if structural conditions are favorable the ore may extend from the 3 to the 100 bed. The bulk of the ore mined at Mascot has come from some part of this 97-ft. zone. A small amount of ore has been developed just above the minus-64 sand. This is, stratigraphically, lower than any other known ore in the district.

### *New Market*

There are two outstanding ore zones at New Market. The better one occurs between the 10 and 28 beds, the other between the 44 and 50 beds. Either or both zones may contain ore, and when the structural conditions are favorable the entire zone from the 10 to the 50 bed is commonly mineable. It is this 40 ft. of formation that has yielded the greatest amount of ore at New Market. At one place, a very small pocket of ore has been mined as high as the 140 bed, the highest stratigraphic level known to have produced ore in the district. No ore has been developed at New Market corresponding to the lower producing zones of Mascot and Jefferson City.

### *Jefferson City*

The 48 to 115 zone has been the main producer at Jefferson City. A fair tonnage of high-grade ore has been mined from the 3 to minus-13 beds.

## PRACTICAL APPLICATIONS

A comprehensive knowledge of the stratigraphy of the district has found practical application in geologic mapping, in prospecting, and in mining.

The commercial ore occupies only a part of the Kingsport formation. Because of the rarity of outcrops of zinc mineralization, surface prospecting is largely a matter of testing stratigraphically and structurally favorable areas. An accurate surface geologic map is a necessity therefore. Surface mapping is difficult because of scarcity of rock outcrops. However, it is made easier by the presence of such useful key beds as the chert matrix and the heavy sand as float blocks in the residual soil.

Underground geologic mapping also requires a thorough knowledge of the local stratigraphy. The minor units, especially the variable sequence of dolomites, cherts and sands above the Grasselli dolomite, are recognized. Smaller units are less easily identified in the more uniform, predominantly calcareous zone below the Grasselli dolomite.

The detection of definite parts of the formation that can be recognized easily in prospect drill holes and in mining has defined ore zones and delimited the stratigraphic extent of the ore bodies. This information has effected significant economies by reducing the underdrilling and overdrilling of both surface and underground prospect holes.

Naturally, key beds are recognized more easily in diamond-drill core than in churn-drill cuttings. The change from medium gray to nearly white dolomite at the minus-C oölite, followed, 80 to 83 ft. below, by the change from the dark Grasselli dolomite to a preponderance of limestone or crystalline dolomite, makes the most satisfactorily identified zones in churn-drill holes

The chert matrix sand and the minus-C oölite are particularly striking in insoluble residues made from churn-drill cuttings, although they are not always readily identified in the untreated cuttings.

Familiarity with the stratigraphy of the mine sections increases the efficiency of mining. The development of mine workings generally is based on the information furnished by surface prospecting. As stopes advance, the strata cut by mining can be compared with those penetrated by near-by drill holes. The knowledge gained by this procedure frequently guides the mining toward prospect drill holes. Similarly, when two stopes are being mined toward each other, the zone mined in one serves to guide the mining in the other. Knowledge of the beds in any stope often supplies a cue to the possible proximity of overlying or underlying ore runs, and familiarity with the habits of the ore in certain zones generally aids in planning developments and directing mining.

## ACKNOWLEDGMENTS

The authors wish to acknowledge the work of the late Mark H. Newman and of Hiram F. Mills in devising the nomenclature used in the mine sections and in measuring and describing an important part of the Mascot section; also to thank the American Zinc Company of Tennessee and E. I. duPont de Nemours and Co. for permission to publish this paper, and Dr. Charles H. Behre, Jr., for able and constructive criticism.

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# Chert in the Kingsport Formation at Mascot, Tennessee

By Arthur T. Allen,* Junior Member AIME

(New York Meeting, February 1948)

## Abstract

The Kingsport formation, a part of the Knox dolomite of Ordovician age, is composed of 538 ft of dolomite and limestone. Numerous bands, layers and nodules of chert, arenaceous and shale zones are present. The characteristics displayed by these features have been useful in mapping, prospecting and mining. Perhaps a better understanding of the origin of these zones will increase their usefulness.

## Introduction

The presence of numerous bands, nodules, and irregular bodies of chert within the Kingsport formation in the Mascot locality has long been recognized. Oder and Miller[1] mention the general characteristics, but do not deal with the chert occurrences in detail.

When the occurrence of chert is studied in fresh exposures underground, a variety of characteristics is noted which has been useful in correlation and in detailing the sub-surface stratigraphy. Although the writer has no desire to enter into a controversy over the origin of chert, many observations have been made of its local characteristics.

A detailed stratigraphic section is included which shows the position of the individual horizons discussed in the following paragraphs.

The term "chertification" has been adopted from Fowler and Lyden[2] for use here. It refers to the replacement of limestone by introduced silicic acid. The writer uses it to refer to the occurrence of introduced or original chert in limestone and dolomite.

In this writing "syngenetic chert" refers to chert which has been deposited contemporaneously with the enclosing limestone or dolomite. Also, it is considered to include interstitial silica shown by chemical analysis and insoluble residues. The term "epigenetic" is applied to all chert which has been introduced after consolidation of the rocks in which it occurs, regardless of the nature of the depositing solutions.

## Syngenetic Chert

It is the belief of the author that a large percentage of the chert found in the Kingsport formation is syngenetic. The widespread occurrence of similar nodules and bands of chert in the same horizon far from one another and far from areas of deformation or alteration of the enclosing rocks is difficult to explain by infiltration of the silica after compaction of the enclosing rocks. When chert of this type does occur near zones of movement it is likely to be involved in faulting and brecciation which indicates that it was deposited prior to deformation.

## Older Epigenetic Chert

There are several occurrences of nodules in the Kingsport which seem to be epigenetic as pointed out in the following examples:

A cherty zone occurs 26 ft above the o bed (Fig 1). This is not always present but is found in the greater part of mine work-

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Manuscript received at the office of the Institute March 26, 1947. Issued as TP 2299 in Mining Technology, January 1948.

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[1] References are at the end of the paper.

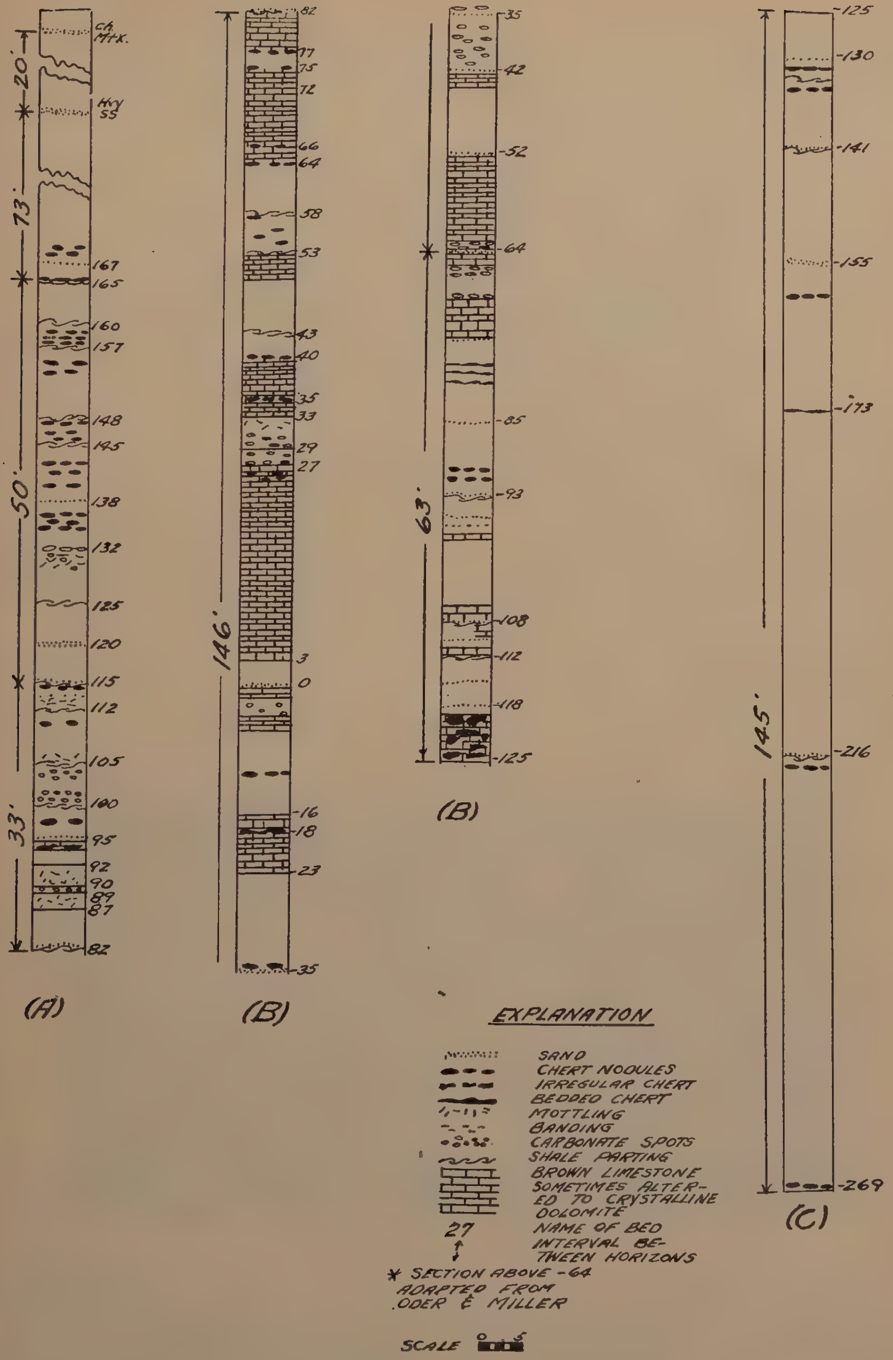

FIG 1—DETAILED SECTION—MASCOT, TENN.

Kingsport FM
(A) Upper Dolomite Member
(B) Midde Limestone Member
(C) Longview Dolomite Member

ings cutting this horizon. The characteristic large blue nodules are frequently oolitic. When present, the oolites are small, irregular in size and shape and vary in color from blue to brown. In numerous exposures these nodules have been noted to include rounded masses of gangue dolomite and brown limestone. The dolomite spots included within the nodules are similar in appearance to the carbonate spots which are scattered throughout the dark gray dolomite and designated the 27–29 zone, which lies immediately above the chert. The limestone inclusions resemble the brown limestone of the 3–26 zone found immediately beneath the chert in unaltered sections (Fig 2).

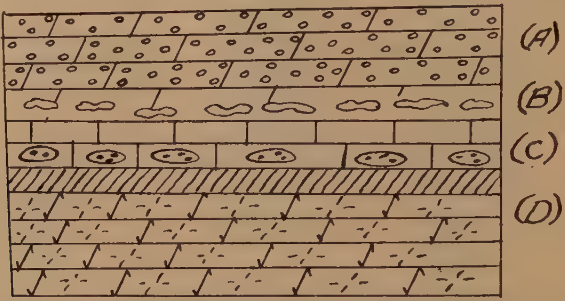

FIG 2—SKETCH OF 26 BED CHERT.

A. 27 Bed, B. Dark brown chert, C. Blue chert including dolomite and limestone, D. Coarsely crystalline dolomite.

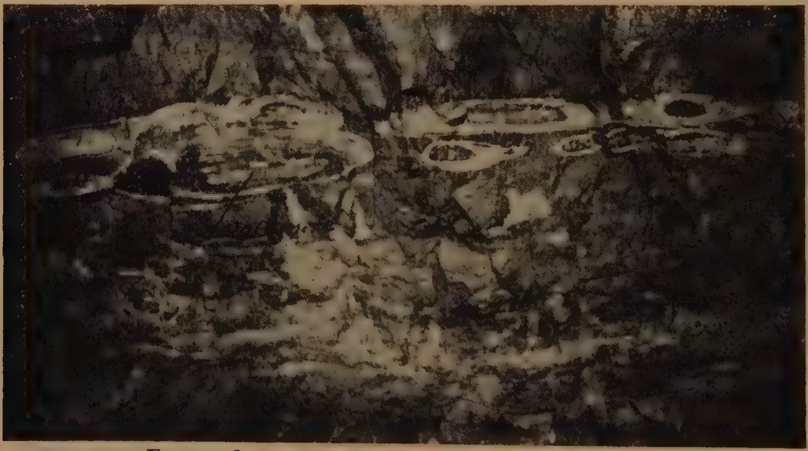

FIG 3—78 BED CHERT SHOWING DOLOMITE INCLUSIONS.

The chert has been deposited later than the compaction of the strata immediately below and above it. The silica infiltration is prior to alteration since the limestone inclusion sometimes occurs in areas where the limestone of the 3–26 zone has been altered to coarsely crystalline dolomite. It is believed that the silica solution has been precipitated along a zone of weakness with fragments of dolomite and limestone acting as nuclei for the formation of chert nodules.

In addition to the occurrence noted above, brown chert nodules occur at various points in the 3–26 limestone section. None of the chert zones are continuous laterally; the nodules often occur near the top, middle, or bottom of the 23 ft-thickness, and are associated with stylolites and shale seams. Within a lateral distance of 100 ft the nodules may disappear entirely and only the stylolite or shale seam is present.

The 78 bed chert occurs near the top of a limestone section 16-ft thick, which is frequently altered to crystalline dolomite. In Fig 3, a small fragment of gangue dolomite may be seen included within the nodules which might indicate an epigenetic origin. However, when following this zone along the strike it is seen to be quite variable and insufficient evidence has been found to state definitely that it is epigenetic.

An occurrence of chert, immediately above the −64 sand, shown in Fig 4, may be indicative of the formation of the epigenetic nodular chert. Within a distance of only a few feet, concentric banding in dolomite changes into dolomite nodules containing chert centers and then into chert nodules as illustrated by sketch.

Two of the more easily identified oolites occurring in the Kingsport formation at Mascot show rather interesting characteristics, which may be construed to be proof of an epigenetic origin. The small ovoid oolites occurring at the −18 horizon have been noted in limestone, whereas, in other localities they are included in brown chert nodules which are quite distinctive. The −C oolite, occurring 164 ft above the 0 sand often shows the same characteristics. Recently, in an extensive prospecting program, almost every drill hole showed the brown concentric oolites in a dolomite matrix, whereas, in other localities the oolites occur in brown chert. Apparently the chert is epigenetic and has replaced the limestone and dolomite, where the oolites are included in chert.

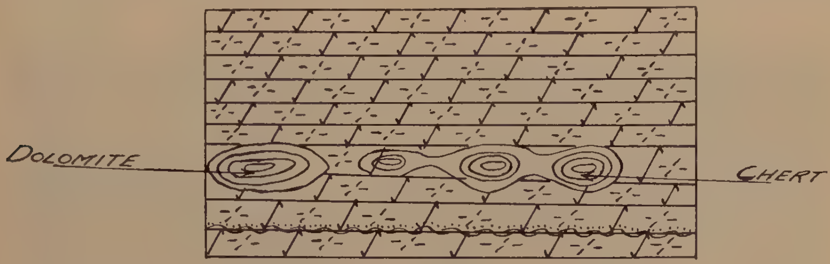

FIG 4—SKETCH OF NODULES ABOVE −64 SAND.

The 53–58 horizon is interesting in the Mascot locality. This bed as seen in Fig 5 is 5 ft-thick and composed of light gray fine-grained dolomite with scattered white carbonate spots. The characteristic white concentric nodules occurring in bands, are practically always present and are scattered throughout the total thickness, although any one band is not continuous over a large area. The nodules roughly parallel the bedding but are not necessarily controlled by it. The shape of the nodules indicates formation in zones of weakness and possibly at intersections of joints. Also, the presence of dolocasts included within the nodules point to an epigenetic origin.

The occurrences of chert listed above may be classified as older epigenetic chert. It is the belief of the author that the silica was deposited in zones of weakness by descending solutions after consolidation of limestone and dolomite and prior to metamorphism and deformation because of the inclusions of unaltered limestone within the nodules occurring in metamorphosed sections.

### YOUNGER EPIGENETIC CHERT

The chert classed as younger occurs associated with zones of faulting and weakness regardless of position in the column and has been noted in the following horizons: −118 zone, −77 sand, −64 sand, (Fig 6) between −35 and −42 sand, −18 bed, 27 bed, 35 bed, 53 bed, 80 bed, and 95 bed.

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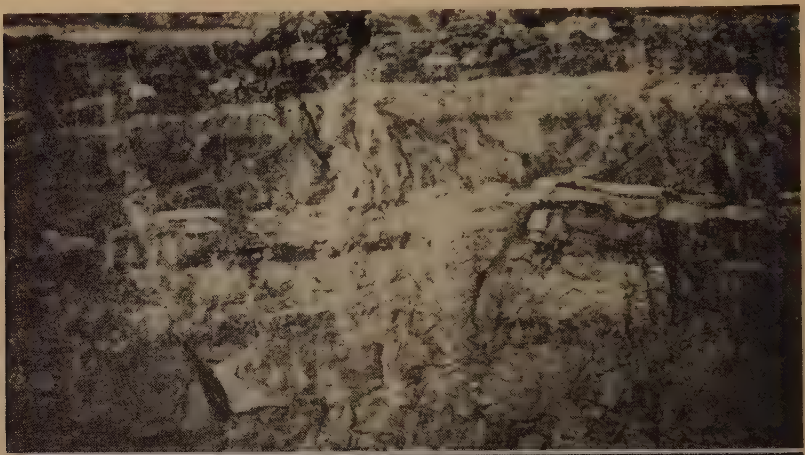

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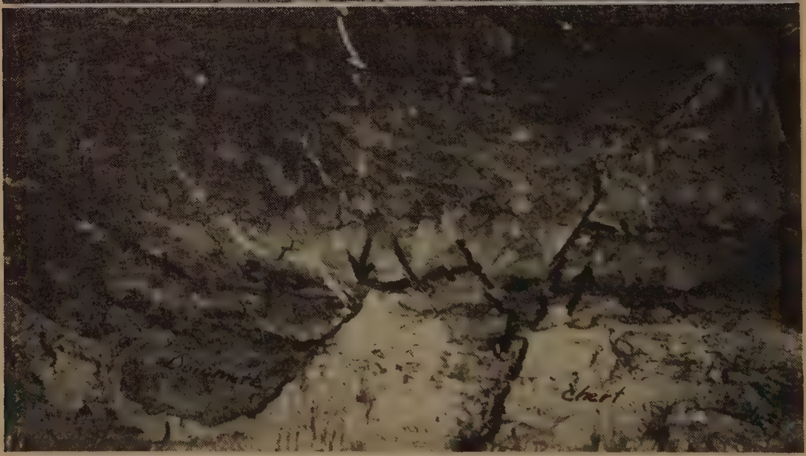


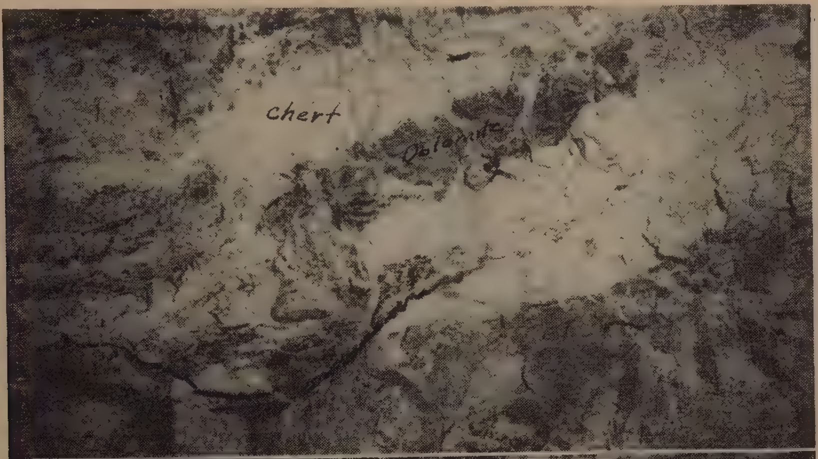
Fig 5—(Top) 53–58 bed containing white chert nodules.
Fig 6—(Bottom) Epigenetic chert below 2A sand.

The younger variety always exhibits the same characteristic whiteness, opaqueness, and irregularity. It is younger than the metamorphism and the deformation but is older than the ore deposition. There seems to be a definite affinity existing between the younger chert and the crystalline dolomite. Wherever it is associated with faulting, it usually seeks the crystalline zones to replace. This is probably caused by the ease with which the larger crystals may be dissolved (Fig 7).

Examples of the younger chert may be pointed out in numerous exposures intimately associated with faulting and grading laterally from the fault in the following sequence: chert, chert and crystalline dolomite, dolomite without any associated chert. The width of the zone of gradation varies from one locality to another, depending on the volume of chert present. This condition observed megascopically is verified by the study of a series of thin sections from samples taken across the zone.

Younger chert occurs in many instances not intimately associated with faults. In the normal sequence no chert is present at the 80 bed horizon, but a shale seam is found. However, at this zone in the western part of the mine there is a large mass of epigenetic chert pinching out laterally into the shale parting. Observation of this locality shows that two periods of secondary chertification are present. The older variety occurs as "fingers" of dark, smoky chert, surrounded by the younger white chert. The fact that the younger chert includes the older, precludes the possibility of the younger being syngenetic. Therefore, since the older variety does not normally occur, it too must be epigenetic. The younger chert is more bountiful than the older where they are associated. This horizon is shown without chert in Fig 8 and containing chert in Fig 9.

In some cases the 35 bed zone shows both younger and older chert. The older variety at this point, however, may be syngenetic. Silicic acid has penetrated this zone, partially dissolving the older, brown



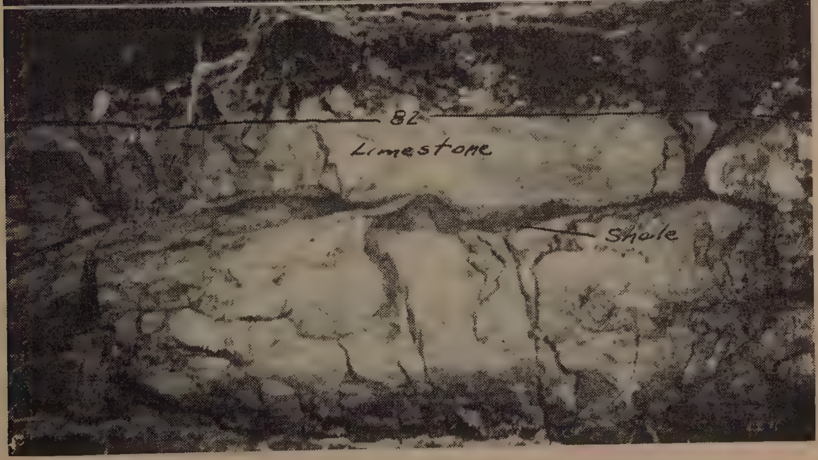


Fig 7—(Top) Epigenetic chert replacing crystalline dolomite.
Fig 8—(Bottom) Normal occurrence of 80 bed.

nodules and forming large irregular masses of younger, white chert surrounding the brown nodules. The epigenetic chert here and in the 80 bed noted above is not directly connected with any major faulting.

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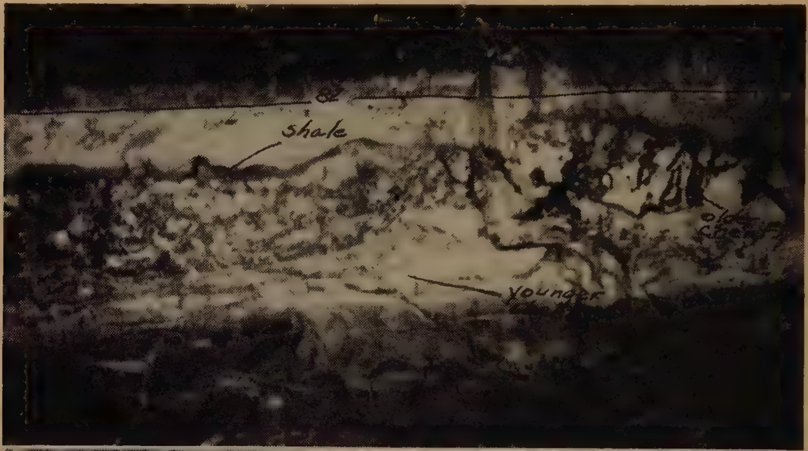

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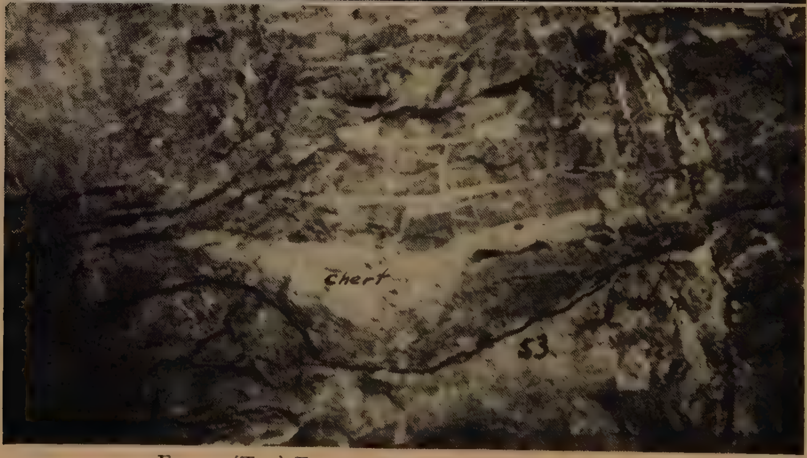

Fig 9—(Top) Epigenetic chert at 80 bed.
Fig 10—(Bottom) Epigenetic chert at 53 bed.

The 53 bed zone is characterized by the presence of a green shale seam separating a bed of dark gray, mottled dolomite below from a zone of light gray, dense, spotted dolomite above. In at least one locality, a mass of white, irregular, epigenetic chert has been noted and is shown in Fig 10. This showing of chert indicates that it has been introduced along the shale seam and has either replaced or deformed the dolomite above it.

Another type of younger chert has been noted in crosscuts from the main level low in the Kingsport formation. Here, adjacent to faults, coarsely crystalline beds have been brecciated and recemented by black opaque chert.

### Surface Residual Chert

The numerous large irregular masses of chert found at the surface along the outcrop of the middle limestone member and the Longview member of the Kingsport are responsible for the misconception of the chert content of the beds. In none of the mine exposures examined have such large pieces of chert been found except for a few occurrences of epigenetic chert.

Chemical analyses made by the American Zinc Co. and studies of insoluble residues indicate considerable interstitial silica which may have been deposited syngenetically in the limestone and the dolomite. It appears likely that concentration by surface weathering may have converted some of this silica into the porous chert masses seen in the residuum along the outcrops.

## CONCLUSIONS

In brief, it is likely that three types of chert occur in the Kingsport formation: 1. Syngenetic chert occurring as nodules and bands. 2. Older epigenetic chert occurring as nodules and fingers, and 3. Younger epigenetic chert associated with faulting and deformation.

It is thought that the first two resulted from silica being deposited contemporaneously or shortly after the other sediments. The third is assigned to a hydrothermal origin, the silica-bearing solutions having ascended along fault planes and having been precipitated in zones of weakness and deformation.

## ACKNOWLEDGMENTS

The writer wishes to express his appreciation to the American Zinc Co., especially to Mr. Harley A. Coy, General Superintendent, for permission to do this work, to Dr. Charles R. L. Oder, Chief Geologist, for advice and helpful suggestions, and to Mr. James E. Ricketts for aid in making underground photographs.

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## DISCUSSION

M. M. BUTLER*—Mr. Allen's commendable paper deserves consideration as it may be the key to the mode of origin relating to various ore deposits. Ortuno[3] suggests that limestones are formed from ascending solutions, which if assumed as replacing the stylobite or shale seam would place Allen's Older Chert as residual while the younger chert is precipitated after the limestone thus causing Allen's picture to become somewhat predictable.

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* Mining Engineer, Casper, Wyo.

[3] *Eng. and Min. Jnl.* (Jan. 24, 1925).

# Mining and Geology at the Helen Mine

By S. J. Kidder* and G. C. McCartney,† Members A.I.M.E.

(Chicago Meeting, February 1946)

## Abstract

The Helen Mine, of the Algoma Steel Corporation, in the Michipicoten district, Ontario, Canada, has produced more than 6,240,290 tons of iron ore. Prior to and during World War I, 2,823,369 gross tons of brown ore were shipped, and from 1939 to the end of the 1944 season, 3,416,921 gross tons of siderite ore were produced from open-pit operations.

Brief descriptions are given of the nature of the ore, the mining methods employed and the manner of treatment of the ore. Shipment of the finished product, known as Algoma sinter, is made to blast furnaces in both Canada and the United States.

The iron-ore bodies—i.e., the oxide ores of the Old Helen and the siderite of the New Helen—are a part of a tabular shaped area of iron formation that extends over a length of more than 10,000 ft. and has a maximum width of 1000 ft. The ores are considered to be a product of introduction of iron-bearing material accompanied by replacement of the rocks already present. It is shown that introduction took place subsequent to the development of the major folded structures. It is believed the iron-bearing material emanated from an igneous source.

## Introduction

The Michipicoten mining district derives its name from the Indian word[1] meaning "place of bold promontories" or "region of big places," and anyone who has visited this part of Canada will agree that the district is well named.

In 1897, there was active prospecting for gold in the district and the Ontario Bureau of Mines[2] opened a recording office at the Hudson's Bay post at Michipicoten River. In the summer of 1897, Ben Boyer, probably searching for gold, discovered an outcrop of brown iron ore at the east end of a small lake, less than 8 miles, in an air line, northeast of the nearest point on Michipicoten Harbour.

In the following year, extensive surface exploration and diamond drilling disclosed a commercial body of nonbessemer brown ore, possibly 700 ft. long and 200 ft. wide, of about the following analysis:[3] Fe (natural), 57.15 per cent; S, dried at 212°F., 0.113; $SiO_2$, dried at 212°F., 4.00; P, dried at 212°F., 0.125.

The property (Figs. 1–3) was taken over by E. V. Clergue and ultimately was transferred to the Algoma Steel Corporation.

The Algoma Central Railway shortly afterward built its line from Michipicoten Harbour to the mine, together with an ore dock at the Harbour. Thus, within three years of its discovery, the mine had adequate shipping facilities and a market for its ore.

## Production of Brown Ore and Pyrite

The first shipment of brown ore was made in July 1900. By 1909, the mine was shipping at the rate of 170,000 long tons

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Presented at 18th annual meeting of the Minnesota Section, Minneapolis, Jan. 15, 1945. Manuscript received at the office of the Institute Sept. 5, 1945. Issued as TP 1971 in Mining Technology, March 1946.

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[1] References are at the end of the paper.

per year, and when mining was stopped in 1918 the total production had amounted to 2,823,369 tons of brown ore and 51,930 tons of pyrite.

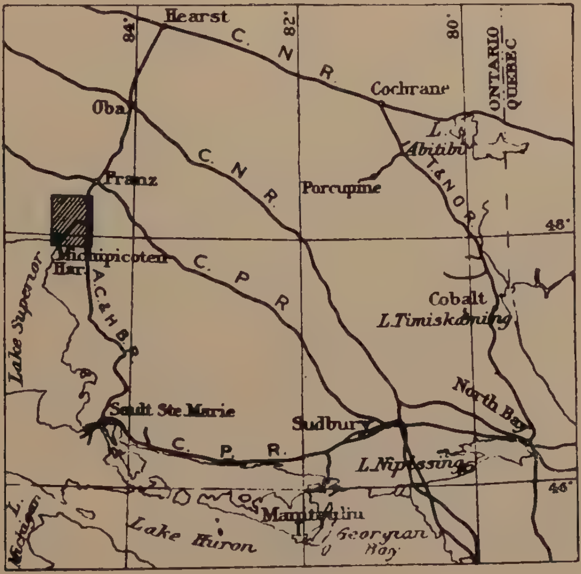

FIG. 1.—MAP OF AREA.

Mining was commenced at the east end of Boyer lake, where the ore is reported to have extended to a height of 100 ft. above the lake. The ore was taken out in benches, using steam shovels loading directly into railway cars.

The lake itself was a rock basin 133 ft. deep. Part of the water was removed by a rock cut at the outlet and the remainder was pumped out.[4]

Open-pit operations were continued for some distance below the level of the lake, after which two vertical shafts were sunk in the country rock on the south side of the ore, one being used as a service shaft and the other for hoisting ore in balance with 2-ton skips.

By means of these two shafts, development was carried to a depth of 651 ft. On the upper levels pillars were left between the underhand stopes and on the two bottom levels the ore was recovered by slicing and caving from sublevels.[5]

On the upper levels the ore body was about 200 ft. long. Though the length and width varied a good deal on the lower levels, the floor area remained fairly constant until the ore body decreased in size near the bottom.

### SIDERITE DEPOSIT

Of particular interest in connection with later operations was the finding of siderite immediately east of where operations were started in Boyer lake.

Before the exhaustion of the brown ore deposit, the Algoma Steel Corporation undertook to explore this body of siderite and between 1913 and 1917 siderite was exposed on the surface in 13 trenches. Widths of ore ranged from 46 to 272 ft. and averaged 165 ft. for a distance of approximately 3000 ft. Below the surface similar ore was cut in 15 diamond-drill holes in the same area. The deepest holes reached a depth of 1700 ft. below the sur-

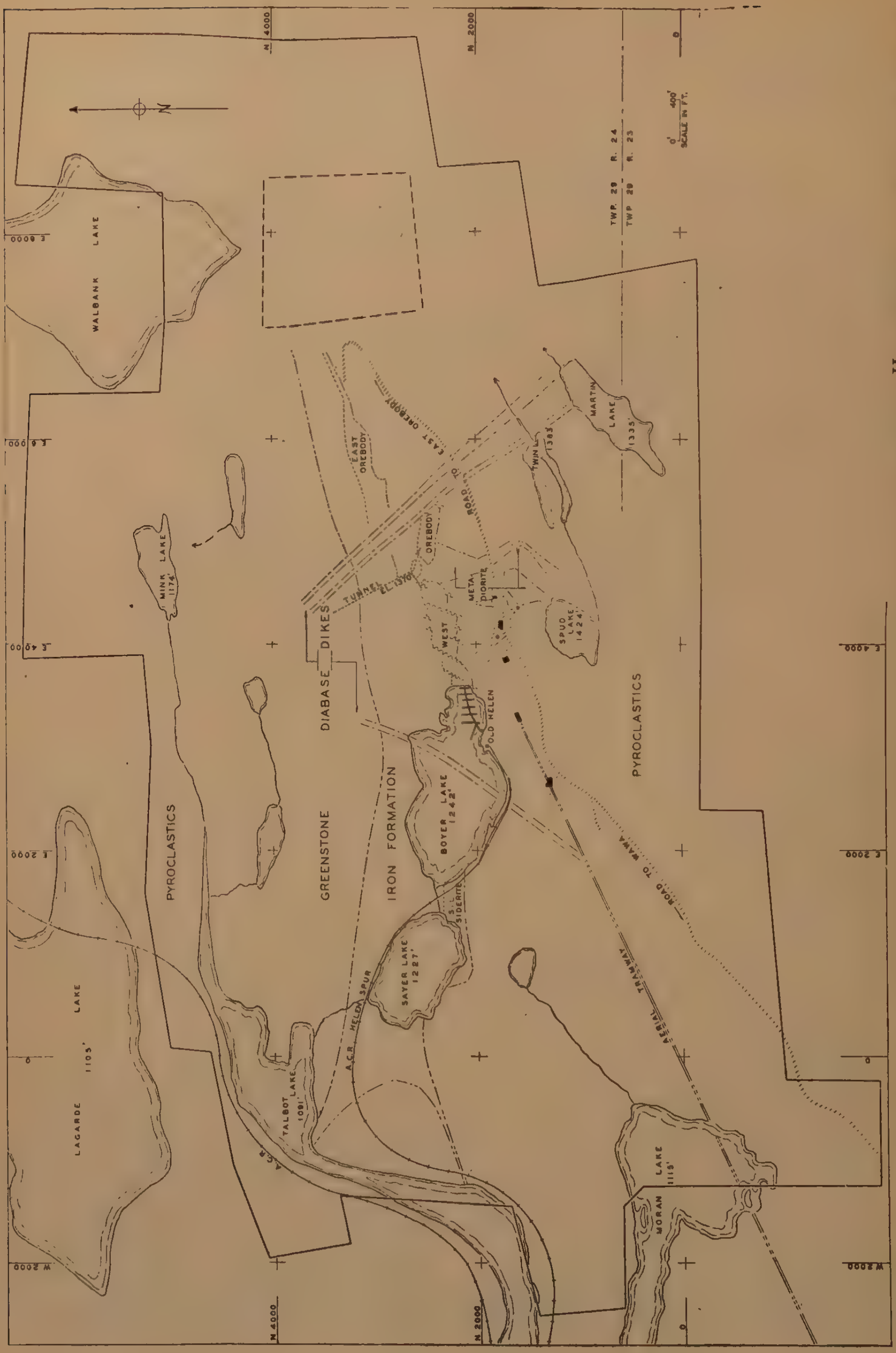

FIG. 2.—PLAN OF PROPERTY SHOWING POSITION OF GEOLOGICAL FORMATION, HELEN MINE.

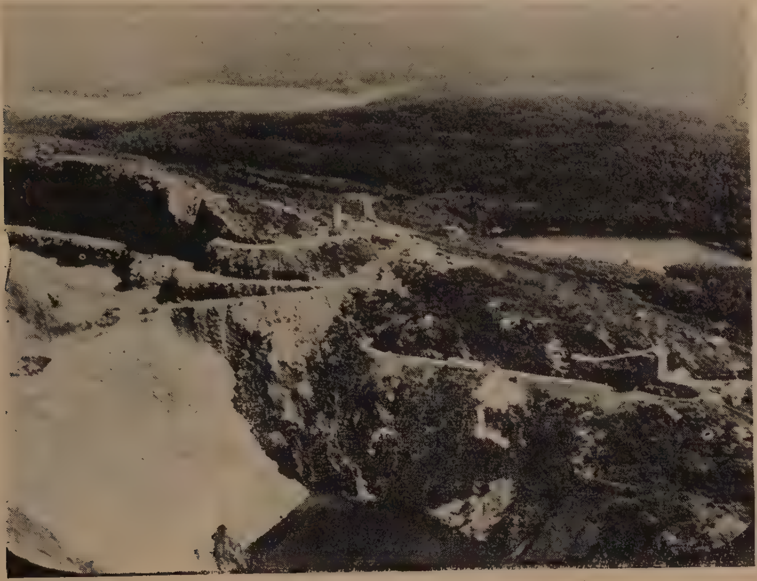

FIG. 3.—HELEN OPEN PIT FROM AIR.

face without finding the bottom of the ore. The drilling and trenching indicated the presence of an ore body containing more than 67 million tons to a depth of 1700 ft. below the top of the hill and to the east of Boyer lake (Fig. 2).

The deeper holes were more widely spaced than those nearer the surface, but they established the continuity of the ore and led to the belief that an estimate of 100 million tons might not be excessive for the whole ore body.

The grade of the ore from the surface to a depth of 324 ft. showed the following analysis:[6]

| Constituent | Low-sulphur Ore, Per Cent | High-sulphur Ore, Per Cent |
|---|---|---|
| Fe (raw ore) | 35.61 | 37.80 |
| S | 1.44 | 7.93 |
| $SiO_2$ | 7.28 | 6.90 |
| Mn | 2.09 | 2.08 |
| P | 0.016 | 0.014 |
| Fe (roasted ore) | 52.03 | 52.36 |

The variation in sulphur was due chiefly to the presence of pyrite on the footwall next to the iron formation.

Between 1913 and 1921, the company had gained much experience in roasting siderite ore from the Magpie mine, 13 to 14 miles north of the Helen. Approximately 1,500,000 tons of that ore had been roasted in rotary steel kilns 125 ft. long and 8 ft. in diameter, using powdered coal as fuel. The finished product from the Magpie mine in 1916 averaged: Fe, 50.10 per cent; S, 0.136; $SiO_2$, 9.14; Mn, 2.74; P, 0.013.

Weighing the various factors that would affect large-scale production from the Helen siderite deposit, the company decided in 1937 to develop the mine and install the necessary plant for a minimum output of 1500 tons of finished sinter per day, equivalent to 300,000 tons for an ordinary shipping season.

The plans finally adopted[7] called first of all for closer drilling and sampling of

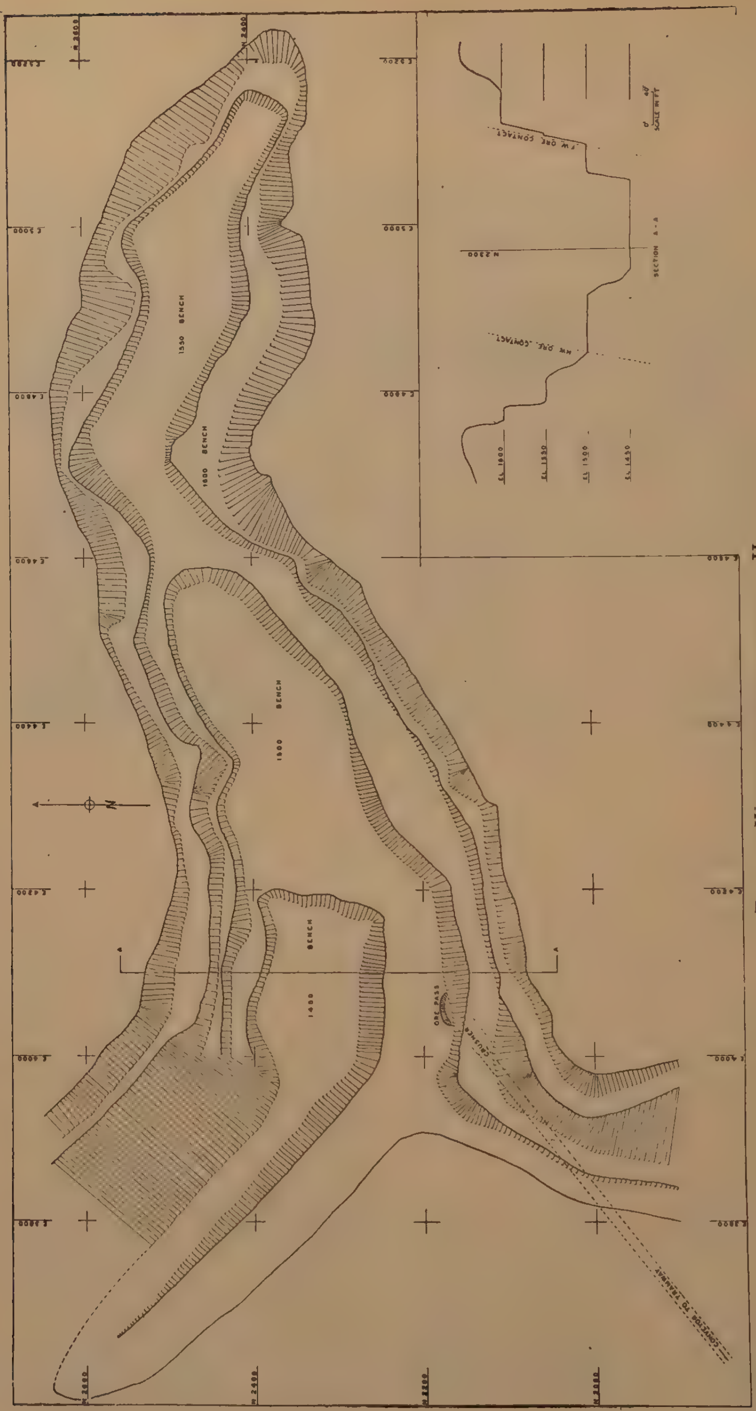

FIG. 4.—WEST ORE BODY, OPEN PIT, HELEN MINE.

the western half of the ore body above the 1450-ft. horizon. This area contained roughly 4 million tons of ore, which the drilling indicated would average: Fe, 35.79 per cent; S, 2.39; $SiO_2$, 4.79; Mn, 2.09.

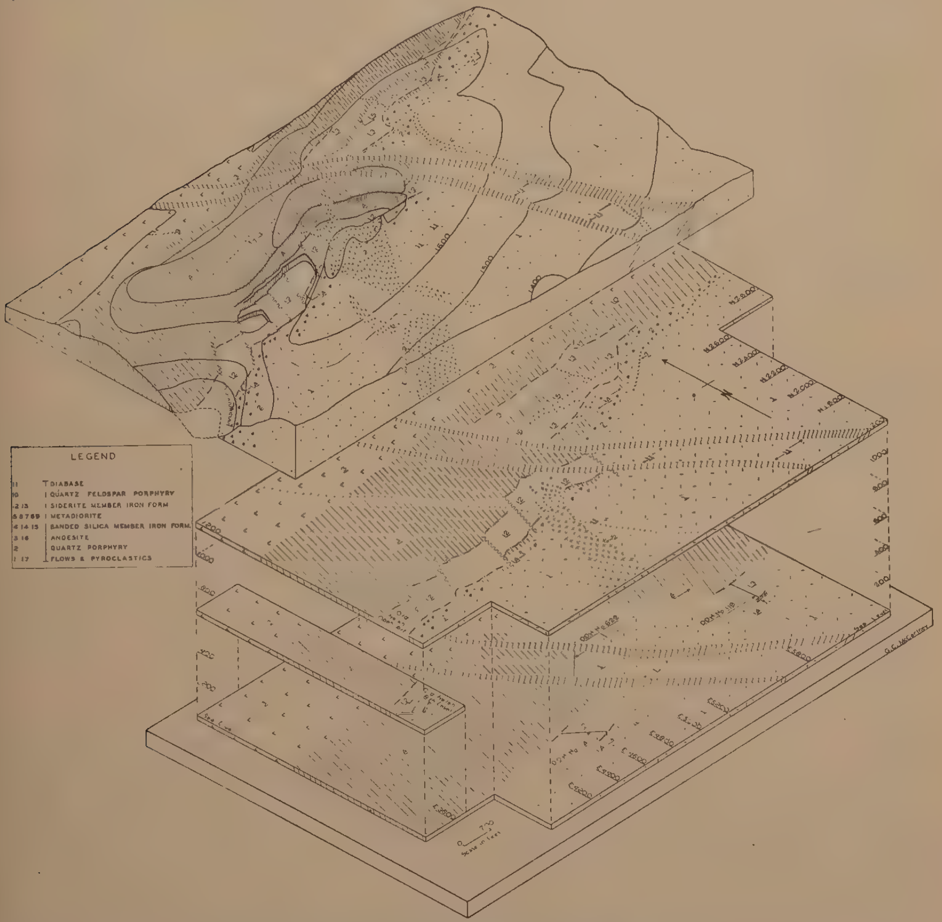

FIG. 5.—BLOCK DIAGRAM, HELEN MINE, SHOWING WEST ORE-BODY OPEN PIT AND STRUCTURAL RELATIONSHIP OF ORE BODIES TO ADJACENT ROCKS.

With a shrinkage of 32 per cent in weight shown in the sintering tests made in the United States, this ore after sintering would average 52.63 per cent iron, 0.03 sulphur, 6.95 silica and 3.03 manganese.

## MINING OPERATIONS

The original layout provided for open-pit operations (Figs. 4 and 5) starting on the 1600-ft. horizon, which would give a maximum height of bank of slightly more than 120 ft. when mining reached the crest of the hill. There was no problem of removal of overburden as the ore outcropped on the surface and the weathering and oxidation of the siderite actually raised the grade slightly.

Much valuable information on blasthole drilling and shoveling at the Tilden pit of the Cleveland-Cliffs Iron Co. was furnished by the officials and engineers of that company. Following their practice, it was decided to begin churn drilling at the Helen with 9-in. holes and to use a 2½-yd. electric shovel for loading the broken ore.

Haulage would be kept at a minimum by dumping broken ore into a raise feeding an underground crusher at elevation 1450. The maximum length haul on the 1600-ft. bench would thus not exceed 1230 ft., making the average length of haul considerably less than 600 ft., as the orepass was near the widest section of the ore body. Diesel tractors and 11-yd. trailers, having a capacity of 15 long tons each, were selected for this part of the hauling.

### *Churn Drilling*

Drilling was started in May 1939 with experienced drillers from Michigan. Local men of ability were put on as helpers and these men in turn became operators, and all subsequent operators have been trained locally.

For the first two seasons all drilling was in ore and one Bucyrus Armstrong Model 29T drill operating two shifts per day was able to keep ahead of the blasting. Later, as the tonnage broken was increased, and drilling for bank waste removal was begun, a second drill of the same type was added.

Spacing of the holes, depth drilled below grade, and manner of loading, have not varied greatly since the work started except where ground conditions varied. The tendency, however, has been to break the ore finer so as to reduce secondary blasting.

In general, holes are spaced 26 to 28 ft. apart, with the distance between rows from 22 to 24 ft. Most holes are drilled from 6 to 8 ft. below grade.

The total footage of 9-in. holes drilled since operations were started now amounts to 96,015 ft., of which 76,463 ft. has been in ore and 19,552 ft. in rock (Table 1). In ore, the average drilled per hour is about 3 ft. The average for all drilling to date has been 2.0 ft. per hour.

TABLE 1.—*Drilling Record*

| Year | Footage Drilled | | | |
|---|---|---|---|---|
| | Ore | Waste | Total | Per Hour |
| 1939 | 8,295 | | 8,295 | 2.85 |
| 1940 | 11,871 | 781 | 12,652 | 3.01 |
| 1941 | 16,376 | 6,331 | 22,707 | 2.04 |
| 1942 | 13,418 | 5,888 | 19,306 | 1.73 |
| 1943 | 13,329 | 4,221 | 17,520 | 1.63 |
| 1944 | 13,204 | 2,331 | 15,535 | 1.98 |
| Total to date. | 76,463 | 19,552 | 96,015 | 2.00 |

In 1943 high-carbon bits were replaced by alloy steel made by the Algoma Steel Corporation at Sault Ste. Marie, according to S.A.E. 4340 specifications. This steel contains 1.5 to 2.0 per cent nickel in addition to small amounts of chromium and molybdenum. It forges well at 1900° to 2000°F. and after being allowed to cool is given a short heat in an annealing furnace at 1700° to 1800°F. It is then given a deep quench in water. No oil quenching is used at the Helen.

Bits are collected and distributed to the drills by a D2 caterpillar tractor with a rubber-tired trailer.

### *Blasting*

In the primary blasting, 75 and 60 per cent Forcite are most commonly used, though often some 90 per cent is used in the bottom of the holes. The top 18 to

TABLE 2.—*Explosives per Ton Blasted*

| Year | Gross Tons Blasted | | | Pounds Explosives per Ton Blasted | |
|---|---|---|---|---|---|
| | Ore | Waste | Total | Primary | Total |
| 1939 | 211,900 | | 211,900 | 0.195 | 0.240 |
| 1940 | 588,544 | 11,594 | 600,138 | 0.220 | 0.272 |
| 1941 | 698,606 | 271,093 | 969,699 | 0.211 | 0.315 |
| 1942 | 729,016 | 457,163 | 1,186,179 | 0.214 | 0.327 |
| 1943 | 680,620 | 328,632 | 1,009,352 | 0.214 | 0.295 |
| 1944 | 726,253 | 173,031 | 899,284 | 0.268 | 0.305 |
| Total to date.. | 3,634,939 | 1,241,513 | 4,876,452 | 0.220 | 0.294 |

20 ft. of the holes is loaded with sand for stemming.

In blasting 4,876,452 tons of ore and waste, the powder used in primary blasting has amounted to 0.22 lb. per ton broken. The total amount of explosives used, including that for secondary blasting, has amounted to 0.294 lb. per ton of ore and waste broken (Table 2).

Reinforced Primacord is used in each churn-drill hole, connected to a trunk line of Primacord, which is detonated by an electric blasting cap. A standard blasting switch, placed in a convenient location, is used for connecting the circuit to 110 or 220-volt lighting circuit.

The face above the broken ore and the walls of the pit are thoroughly scaled after each blast. No serious trouble from high banks has been experienced even when the banks reached a height of over 120 ft., as on the first bench. The next three benches below the 1600-ft. elevation have been opened at 50-ft. intervals (Fig. 3).

Safety regulations at the Helen prohibit drilling on top of broken muck piles or near the toe of a steep muck pile. The shovel must be used to place the large pieces or ore or waste in a safe place for drilling, or they may be sand-blasted first and then moved to a safe place by the shovel.

### *Shoveling*

Two 2½-yd. electric shovels are in use, a type 4101 Marion with which operations were started, and a 54B Bucyrus-Erie purchased in 1941, after plans for bank waste removal had been completed.

The total amount of ore and waste moved by the two shovels has been 4,881,448 gross tons. The ore loaded per shovel hour has averaged 135 tons. The average for both ore and waste has been 124 tons per hour.

### *Crushing and Conveying*

The orepass from the underground crusher chamber on the 1450-ft. elevation was driven through to the surface on a 55 per cent grade and, except on the first, or 1600-ft. bench, no grizzly has been used at the dump.

The underground crusher room is equipped with a No. 9 Ross feeder and a 48 by 60-in. Traylor crusher, discharging onto a 30-in. belt conveyor. Dust control is affected by enclosing the chute feeding the conveyor and drawing 2500 cu. ft. of air per min. down through the crusher and exhausting to the surface. Water sprays also are provided at the mouth of the crusher for use at any time except in freezing weather.

When set for a maximum opening of 4½ to 5 in., the crusher handles an average of 170 to 180 tons per hour.

The 30-in. mine conveyor has a total length of 914 ft. and operates at a speed of 224 ft. per min. It discharges into a steel bin, used for loading the buckets of a 3-mile aerial tramway.

The bin is covered and hoods are placed over the loading chutes used for filling the buckets. An exhaust fan and dust collector give satisfactory control of the dust.

### *Tramway*

The tramway (Fig. 6) is a double-rope, continuous type, using 1¾-in. locked coil cable for the loaded buckets and 1¼-in. locked coil cable for the returning empty buckets. The continuous traction rope is a ¾-in. 6 by 7 hemp-center Lang-lay cable 29,000 ft. long with a counterweight at the discharge end of the line for maintaining proper tension on the cable.

The elevation of the terminal where the buckets discharge is 450 ft. lower than the point of loading. There are 32 through-type steel towers, on which the track cables are supported by means of rocking saddles.

The drive is through an 8-ft. grip sheave in the loading terminal, connected by intermediate gearing to a 75-hp. motor. By this means, the speed of the traction rope is kept at 500 ft. per min. At this speed the tramway delivers 135 tons per hr., or 3216 long tons per day.

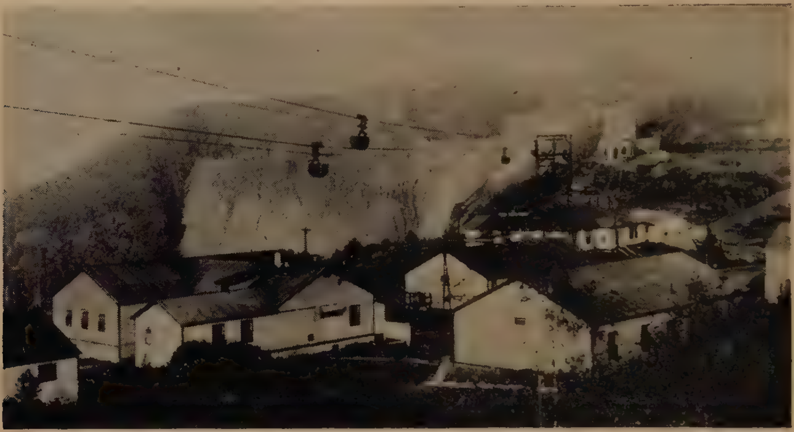

FIG. 6.—CAMP BUILDINGS AND TRAMWAY, WITH OPEN PIT IN BACKGROUND.

The buckets are dumped either directly into a terminal bin adjacent to the sinter plant or onto a stock pile between the terminal bin and the first tower of the tramway.

### *Stock-piling Ore*

Buckets are dumped automatically onto the stock pile by a movable tripper, which rides on the 1¾-in. track cable and is held down by movable anchors on either side of the pile. As dumping proceeds, a 1½-yd. clamshell bucket, with a 60-ft. boom, is used to spread the ore on either side of the centerline of the tramway.

For recovering the ore from the stock pile, there is a reinforced concrete conveyor gallery immediately below the surface, with draw points every 20 ft. A 30-in. belt conveyor extends along one side of the tunnel, and ore can be drawn from any of the 20 draw points by means of an adjustable feeder, which usually is set to deliver 200 tons per hour to the fine-crushing and screening plant.

When first starting to pull ore from the stock pile in the spring, all holes are usually filled with frozen ore. By using a unit heater at the back end of the conveyor gallery and directing the heat up one draw point at a time, little difficulty is experienced in getting the ore to run.

After the center core of the pile is drawn out, the sides are generally knocked down with a bulldozer and then the clamshell used in spreading is replaced by a 1½-yd. shovel, which casts directly into the draw points feeding the conveyor. Even in cleaning up the outer edges of the pile farthest from the conveyor, one tractor and 11-yd. Athey wagon working with the shovel can reclaim the ore at a rate of 200 tons per hour.

TABLE 3.—*Stock-pile Data*

| Year Ending | Placed on Stock-Pile, Gross Tons | With-drawn from Stock Pile, Gross Tons | Stock-pile Expense | |
|---|---|---|---|---|
| | | | Total | Per Ton With-drawn |
| April 30, 1941 | 209,869 | 115,976 | $ 4,894.09 | $0.0422 |
| April 30, 1942 | 437,695 | 412,807 | 14,002.19 | 0.0339 |
| April 30, 1943 | 165,754 | 179,320 | 9,737.95 | 0.0540 |
| April 30, 1944 | 214,785 | 219,857 | 13,077.60 | 0.0595 |
| Total.... | 1,008,103 | 927,960 | $41,713.83 | $0.0450 |

Over a period of four years, during which time 1,008,103 tons was placed on the stock pile, and 927,960 tons was withdrawn, the total stock-pile expenses amounted to 0.045 cents per ton of ore withdrawn from the pile (Table 3).

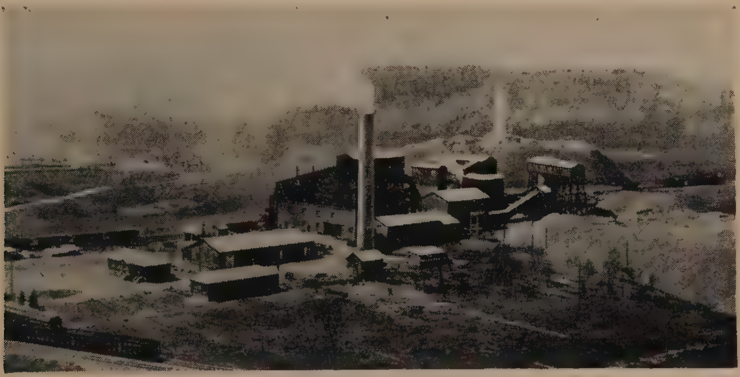

FIG. 7.—SINTER PLANT.

## *Sinter Plant*

In order to better follow the flow of the raw ore through to the finished product, a brief outline of the sintering process is given.

Ore drawn from the tramway terminal bin or from the stock pile is fed over four 4 by 8-ft. double-deck Tyler screens. The oversize from the top screens goes to a 5½-ft. Symons cone crusher and the oversize from the second screen goes to a set of 72 by 18-in. Traylor rolls. The product from both crushers is returned with the raw ore to the Tyler screens until it is all crushed to minus ¼ inch.

Coke breeze from the company's plant in Sault Ste. Marie is brought up from Michipicoten Harbour in hopper-bottom cars, partially dried and crushed to minus ¼ inch.

The main sinter-plant building (Fig. 7) contains three 60-in. by 77-ft. Dwight-Lloyd sintering machines, in conjunction with which there are three 500-hp. fans housed in an adjacent fanroom.

The screened ore is thoroughly mixed with an average of 2.5 per cent by weight of coke. The rubber-lined pug mills in which the mixing is done feed directly onto the sintering machines, building up a bed 13½ to 14 in. deep.

A small oil-fired muffle furnace on top of each machine ignites the charge while the grates are traveling along at a speed of 40 to 45 in. per min. The muffle furnaces have been gradually reduced in size, so that with those at present in use the oil consumption does not exceed one Imperial gallon of fuel oil per ton of sinter produced.

Sinter that will pass through a ½-in. grizzly is returned to the charge bins. It constitutes from 15 to 18 per cent of the feed to the machines. All sinter above ½ in. in size drops into a covered loading bin from which the hopper-bottom railroad cars load for transporting the sinter to the docks at Michipicoten Harbour.

The sides and tops of the cars are sprayed with water during loading but the cargoes seldom average more than 1.5 to 2.0 per cent moisture.

A good deal of attention has been paid to dust control in all of the buildings at the sinter plant. Chutes underneath the crushers are enclosed, permitting 2500 cu. ft. of air per min. to be drawn down through each crusher and exhausted into a dust collector. Points of transfer on the conveyors are also enclosed and the dust exhausted into a dust collector, from which it is returned to the circuit.

There is also an exhaust fan at the discharge end of each sinter machine, just above the place at which the sinter falls into the covered loading bin. Fans are used on the burner floor for bringing fresh air into the building while at other points they are used for exhausting the air from the building.

### *Sampling*

The ore is sampled at the loading terminal at the mine and regular samples of the crushed ore are taken at the sinter plant. Samples of the finished sinter are taken from the tops of the cars each shift, and from the conveyor at the Harbour while boats are being loaded.

TABLE 4.—*Tonnage and Grade of Raw Ore and Sinter*

| Year | Finished Sinter, Tons G | Raw Ore Charged, Tons G | Shrinkage, Per Cent |
|---|---|---|---|
| 1939 | 111,315 | 166,476 | 33.13 |
| 1940 | 361,394 | 524,410 | 31.09 |
| 1941 | 462,075 | 669,351 | 30.97 |
| 1942 | 486,413 | 702,913 | 30.80 |
| 1943 | 450,973 | 652,929 | 30.93 |
| 1944 | 482,083 | 700,842 | 31.21 |
| Total... | 2,354,253 | 3,416,921 | 31.10 |

Average Analysis, Raw Ore, Per Cent

| Year | Fe | S | $SiO_2$ | Mn |
|---|---|---|---|---|
| 1939 | 36.40 | 1.90 | 4.70 | 2.12 |
| 1940 | 36.34 | 2.18 | 5.33 | 2.10 |
| 1941 | 35.13 | 1.85 | 7.38 | 2.05 |
| 1942 | 35.85 | 2.17 | 6.88 | 1.96 |
| 1943 | 35.70 | 2.01 | 6.72 | 2.02 |
| 1944 | 35.74 | 2.37 | 6.40 | 1.91 |

Average Analysis, Sinter, Per Cent

| Year | Fe | S | $SiO_2$ | Mn | P | $Al_2O_3$ | CaO | MgO |
|---|---|---|---|---|---|---|---|---|
| 1939 | 53.40 | 0.028 | 7.16 | 3.02 | 0.025 | 1.73 | 4.15 | 7.53 |
| 1940 | 52.90 | 0.040 | 7.70 | 2.99 | 0.034 | 1.86 | 3.45 | 7.98 |
| 1941 | 50.80 | 0.072 | 10.51 | 2.95 | 0.019 | 2.00 | 3.82 | 7.39 |
| 1942 | 51.53 | 0.077 | 9.93 | 2.83 | 0.019 | 2.52 | 3.77 | 7.22 |
| 1943 | 51.36 | 0.073 | 9.50 | 2.92 | 0.019 | 2.04 | 3.52 | 7.97 |
| 1944 | 51.94 | 0.054 | 9.35 | 2.82 | | | | |

## PRODUCTION AND GRADE

Since the beginning of operations in 1939, the total production of finished sinter to the end of the 1944 season has amounted to 2,354,253 long tons. The raw ore charged during the same period was 3,416,921 tons, indicating a shrinkage of 31.1 per cent (Table 4).

## POWER

All power used in the district is furnished by the Great Lakes Power Co., whose generating plants are on the Michipicoten and Montreal Rivers. The power is transmitted to both the mine and sinter plant at 33,000 volts and then distributed at 2300 and 450 volts as required. Synchronous motors are used on the mine compressors and help materially in maintaining a high power factor.

## GEOLOGY

The geology of the iron deposits of the Michipicoten area has been studied and reported upon at various times since early in the present century. A. P. Coleman and A. B. Willmott[8] reported on the Michipicoten iron region in 1902. The most comprehensive report on the area is that prepared by W. H. Collins, T. T. Quirke, and Ellis Thomson,[4] published in 1926. Since that time, there have been reports on various portions of the area by E. S. Moore,[9] Brown and Morrison,[14] and a number of others.[10-12]

All the rocks of the area, except the younger intrusives, are exposed today in the form of truncated limbs of folds with east-west strike and steep dip to the south. The rock formations are:

Diabase
Quartz feldspar porphyry
Siderite member of iron formation
Metadiorite
Banded silica member of iron formation
Andesite
Quartz porphyry
Flows and pyroclastics; "Wawa tuffs"

In the main, the oldest rocks comprise a series of volcanics made up of flows and abundant pyroclastics (area 1, Fig. 5). These rocks, named the "Wawa tuffs" by Coleman and Willmott,[8] consist of sericitic, chloritic, ottrelitic and carbonatized schists in which "eyes" of quartz are abundant. One horizon in the series is worthy of particular mention for it flanks the siderite ore body (area 2, Fig. 5) along the south side. This horizon is a quartz porphyry in which the "eyes" of quartz are so abundant that the rock is made up of 75 per cent quartz and the remainder sericite and carbonate. Scattered throughout the quartz porphyry are many angular rock fragments of quartz porphyry, which are very similar in appearance to the host rock, and there are present many other fragments of both light and dark-colored schist.

Within the pyroclastic series of rocks described there are well-defined horizons of andesite. The andesite is typical greenstone and sometimes exhibits excellent pillow structures (area 3, Fig. 5). Also within the pyroclastic series, and very often, as at the Helen mine, along the contact between pyroclastic rocks and well-defined greenstone flows, iron formation is present (area 4; Fig. 5). Since the Helen ore bodies are part of the iron formation, a more complete description is given in a later part of this paper.

Definitely younger than the pyroclastics are irregular, dikelike and sill-like bodies of basic igneous material, which in this paper is referred to as metadiorite (area 5, Fig. 5). The rock is made up of chlorite, which is present in clusters of dark green material in a lighter colored chloritic and carbonatized groundmass, giving the rock a patchy appearance. A light green chlorite having the appearance of fuchsite is sometimes in evidence. Iron carbonate crystals are usually abundantly developed throughout, and on surface the rock is characterized by a pitted appearance due to the weathering of the carbonate crystals. The shape of the surface outcrops of metadiorite suggests an intrusive relationship with the enclosing pyroclastics, but in detail it is usually difficult to recognize an intrusive contact. The nature of the origin of metadiorite is best illustrated in area 6, Fig. 5. This outcrop consists of a concentration of chlorite crystals or, i.e., metacrysts of chlorite of the ottrelite and chloritoid variety sufficient to almost completely, but not quite, destroy and replace the older sericitic schist. The metacrysts are oriented at various angles to the schistosity of the older rock. This phase of metadiorite grades into massive metadiorite, in which all evidence of the original pyroclastic rock has been destroyed. It is apparent the metadiorite rocks, at certain times at least, have invaded the pyroclastics via a replacement process.

Another rock type has been recognized, which is similar in appearance to the quartz porphyry already described, except that in this instance the groundmass is red in color. Feldspar is more abundant in this porphyry than in the one described previously. This porphyry is thought to represent a relatively late phase of intrusion.

The youngest rocks in the area are diabase dikes (area 11, Fig. 5). For the most part they are parallel structures with northwest-southeast trend. These dikes dip westward at a steep angle. The more easterly one dips at an angle of 80° and the adjacent one at an angle of 71°. The only dike in the vicinity that trends contrary to this direction is one encountered in the underground workings of the Old Helen mine, which trends northeast-southwest across the property as shown in Fig. 2. This dike dips eastward toward the others at an angle of about 60°.

## Structure of Iron Formation and Associated Siderite Ore Bodies

An outline of the areal extent of the iron formation is given in Fig. 2. The entire iron formation member is a tabular-shaped area which extends over a length of more than 10,000 ft., and has a maximum width of 1000 ft. The west end of the iron formation is divided into two narrow prongs, which gradually diminish in width westward, and the zone actually comes to an end under Talbot Lake. The east end narrows to a width of 300 ft. at the east end of the property. Collins and Quirke[4] have shown that the zone pinches out a few hundred feet east of Walbank Lake.

The average strike of the entire iron formation member is east-west, but in detail it may be noted that from Boyer Lake the iron formation strikes a few degrees north of west toward the west and a few degrees north of east toward the east.

The iron formation consists mainly of quartz, siderite and chlorite. These minerals are present in alternating, essentially parallel bands. The quartz consists of a number of varieties and the varieties in part at least, reflect stages in the development of the entire banded silica member; there is a dark blue variety of quartz that possesses a parallel banding on a minute scale; there are cherty phases of quartz; there are milky quartz bands and there are sugary quartz bands. The bands of quartz range in width from 1/16 to 2 in. and probably average between 1/4 and 1/2 in. The individual bands may be continuous for only a few inches or they may extend a few feet. Collins and Quirke[4] aptly described the banded silica when they reported "the individual laminae are not uniform and laterally continuous. Many of them are lenticular and the banded silica member is in part at least an imbricated succession of discontinuous layers and lenses."

The siderite member of the banded iron formation forms ribbons or bands comparable in width to the silica bands. There is one essential difference, however. The siderite, in addition to forming many parallel bands, which alternate with the silica bands, also fills all cracks and interstices within and between the banded silica, and thus siderite forms a matrix for all brecciated banded silica fragments.

In certain areas chloritic schist is an important constituent of the iron formation and when present it too forms parallel bands that alternate with the quartz and siderite bands.

A very common feature of the banded silica member is the presence of zones of brecciated quartz, which may attain a width of 50 ft., as along the contact between the main banded member and the siderite ore bodies, or they may be present over a width of from 1 to 5 ft. in parallel fashion, with strike similar to the unbrecciated portion of the silica member. The brecciated zones are repeated every few feet. In these instances, the silica bands have been broken into angular fragments, which range in size from 1/4 in. in diameter to slabs of banded silica 2 in. wide and 1 ft. long.

Figs. 2 and 5 show that the main siderite portion of the iron formation is an area along the south side of the iron formation. In the section between Boyer Lake and the two large diabase dikes (the West ore body) a length of 1600 ft. and a maximum width of 250 ft. of siderite is present (area 12, Fig. 5). In the section east of the two dikes (the East ore body) a length of 1200 ft. and a width comparable to the West ore body is present (area 13, Fig. 5).

There has been investigation of the ore

bodies at depth by diamond drilling. Two of the deepest intersections are shown in Fig. 5. Hole No. 114 intersected the West ore body at an average elevation of 200 ft. above sea level; i.e., 1500 ft. below the top of Helen Mountain. This intersection shows that the width and grade of the ore body compare very favorably with that being mined. Hole No. 118 intersected the East ore body at an average elevation of 300 ft. below sea level; i.e., 2000 ft. below the top of Helen Mountain. Again the width and grade of the ore body compare favorably with the ore as exposed on surface.

Almost invariably along the south wall of the ore bodies there is present a narrow band of iron formation, which may vary from a width of a few inches to 15 ft. (area 14, Fig. 5). In general this band resembles the main iron formation to the north of the siderite ore bodies except that here usually there is more evidence of the presence of tuffaceous beds and locally the bands of silica and siderite may be entirely absent or as much as one foot apart with the intervening rock consisting of tuffaceous material. Occasionally too a few feet of siderite is present on the south side of the south contact iron formation, and in such places the contact with the pyroclastics is gradational.

A feature of the East ore body is a band 50 ft. wide of brecciated and at times slightly banded silica, which occupies a position almost in the center of the ore body and trends parallel to the walls of the ore body throughout practically its entire length. This band consists mainly of quartz with siderite present in a network of seams and stringers throughout (area 15, Fig. 5).

## Mode of Origin of the Siderite

A complete structural picture of the ore bodies can be obtained only when a study is made of the mode of origin of the siderite.

It is believed that the siderite was introduced by iron-bearing emanations from an igneous source and that the introduced material has replaced the preexisting rocks. In some places there has been introduction of quartz along with siderite but fortunately the ratio of quartz to siderite has been for the most part quite small. It is believed also that the process of introduction took place subsequent to the development of the major folded structures as seen today.

It has been recognized by Collins and Quirke[4] that "the siderite member of the Michipicoten iron formation is ordinary volcanic rock of the Keewatin schist-complex that has been converted more or less completely into carbonate by a process of chemical replacement." Collins and Quirke[4] believed the siderite replaced the volcanic rocks adjacent to the original surface by uprising hot waters while the rocks were flat-lying. Grout[16] regards carbonate replacement on a large scale as a characteristic of at least moderate depths.

There are a number of features, enumerated below, which indicate that pyroclastic rocks, apparently very similar to some members of the pyroclastic series as seen today to the south of the ore body, represent the original rocks that were replaced by carbonate.

The evidence supporting the conception that the replacement occurred subsequent to the folding of the rocks is mainly structural and has to do with an interpretation of the relationship that exists between the metadiorite and the adjacent rocks.

Probably the most difficult feature to explain in the matter of origin of the siderite is the origin of the very narrow ribbons or bands of siderite within the banded silica of the iron formation. Since the siderite that makes up the ore bodies and the siderite that is adjacent to the ore bodies but is interbanded with quartz is similar in appearance and composition, it is logical to expect it to have a common origin.

In the following paragraphs evidence is presented: (1) that the siderite ore bodies were formed by replacement; (2) that the replacement occurred after the major folding of the rocks; (3) that siderite has been introduced into the banded silica member.

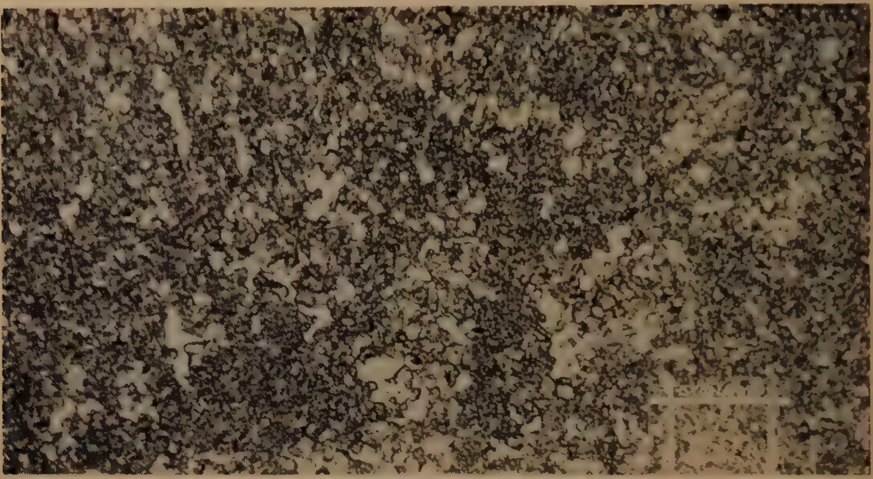

FIG. 8.—SIDERITE ORE FROM WEST ORE BODY.
White areas represent grains of quartz.

### *Siderite Ore Bodies Were Formed by Replacement*

This relationship is illustrated on the surface, especially in the south walls of the ore body where mining operations have not disturbed the rocks. Surface weathering has changed the siderite to the oxide form in a film ½ to 1 in. thick and the areas of brown oxide are readily distinguishable from those in which relatively less siderite is present. Along the entire length of the south side of the ore body, in an almost continuous zone that is usually 20 to 30 ft. wide but may be more, as along the south side of Sayer Lake, siderite has been introduced into the rocks and has selectively replaced the rocks in such manner that a relatively fine-grained groundmass has been replaced, and at the same time the large fragments of schist and quartz porphyry have not been replaced or have been only slightly replaced.

Along the north side of the iron formation member very little siderite appears to have entered the adjacent rocks—in this instance, the greenstones; nevertheless, here and there a trace of oxidized siderite may be recognized in the form of tiny stringers, which extend a few inches into the greenstone member.

Within the ore body excellent evidence of the replacement nature of the ore is afforded by the presence of remnants of unreplaced or only partially replaced material. Within the East ore body, two areas are indicated in which the rock present is identical with the rocks known to be present both to the north and to the south of the ore body. Adjacent on the east side of the east diabase dike, there is one area of considerable size of typical greenstone schist (area 16, Fig. 5). Again, in the contact between the ore body and the banded silica member to the north, near the east end of the East ore body, there is present an area of sericitic and tuffaceous rocks, which are identical to those present to the south of the ore body (area 17, Fig. 5). These rocks represent original unreplaced remnants of the pyroclastic series of rocks.

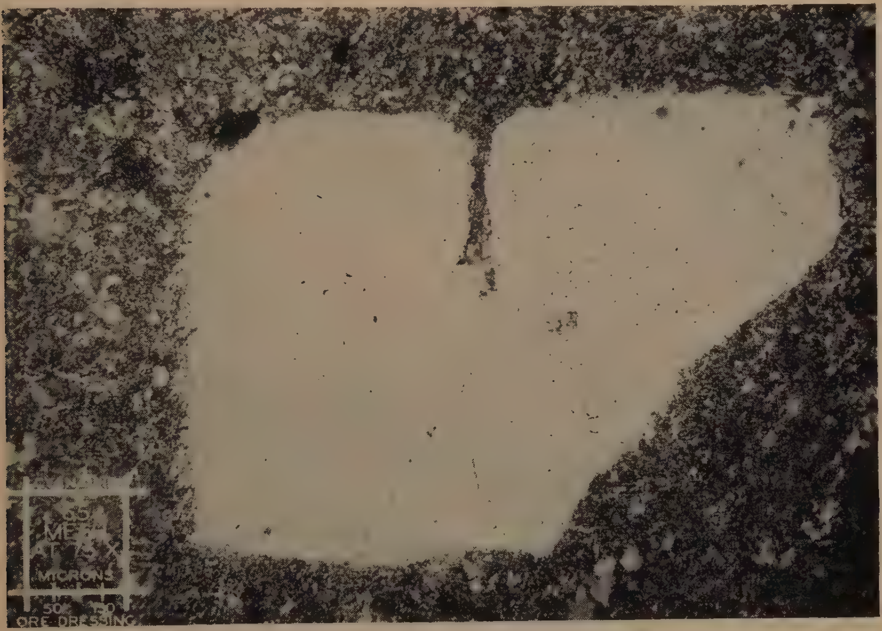

FIG. 9.—PYROCLASTIC WALL ROCK ADJACENT TO SIDERITE ORE BODY.
Large grain of quartz in groundmass of fine quartz and sericite, with a little carbonate.

Another feature of the siderite ore is the presence of zones of relatively large "eyes" of quartz—"eyes" in the order of 740 microns in diameter in a matrix of grains of siderite and quartz in the order of 40 microns in diameter and even smaller. These zones of relatively large quartz "eyes" tend to form east-west bands parallel to the trend of the ore body. The quartz "eyes" identify certain horizons within the siderite ore body as being very similar to some of the "quartz eye" zones noted to be present within the pyroclastic rocks outside of the ore body. The similarity is so great that again it is apparent that the siderite ore body represents the replacement of rocks similar to those that flank the ore body.

The accompanying micrographs (Figs. 8 to 10) illustrate the nature of the siderite ore and the great similarity in appearance between ore and only partially carbonatized wall rocks.

Fig. 8 is a micrograph of a specimen of siderite ore. The irregular white areas are grains of quartz and all other material is siderite. The chemical analysis of this material shows the specimen to consist of 35 per cent Fe and 11.7 per cent $SiO_2$.

Fig. 9 shows a specimen of wall rock. The specimen was taken from a point a few feet south of the south contact of the siderite ore body within the quartz porphyry member of the pyroclastic series. The micrograph shows a relatively large grain of quartz surrounded by a groundmass of very fine quartz and tiny pinpoint areas that represent sericite. There is also some carbonate scattered through the groundmass. It is estimated that three fourths of the rock consists of quartz and the remainder is sericite with a little carbonate. The frayed edges of the large quartz grain plus the peninsula of groundmass material within the grain suggest some reaction between the groundmass and the large grain of quartz.

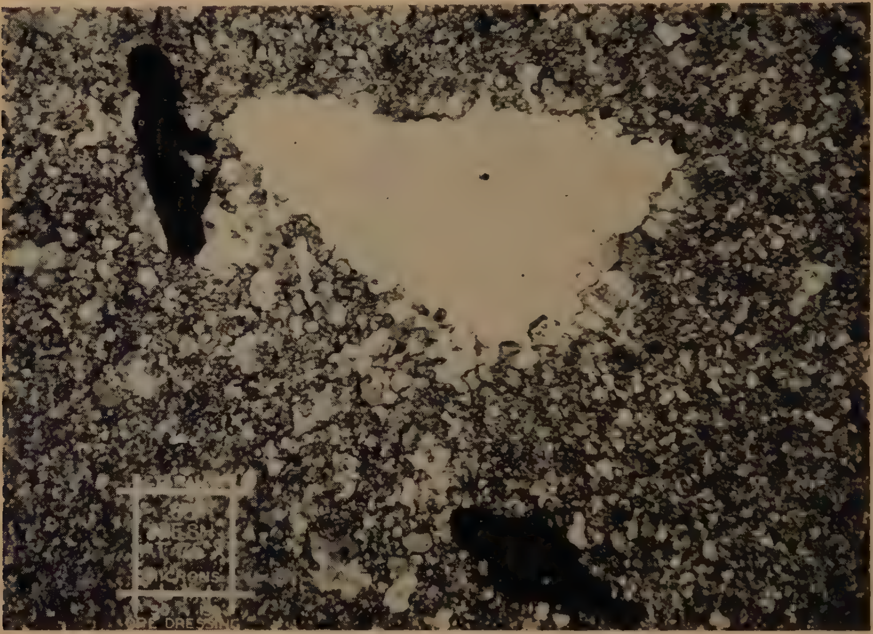

FIG. 10.—SIDERITE ORE CONTAINING QUARTZ "EYE."
Specimen taken from drill core D.D.H. No. 610, at 150 feet.

Fig. 10 shows a specimen of siderite similar to that illustrated in Fig. 8 except that in this instance there is present a large grain of quartz similar to that illustrated in Fig. 9. A comparison of Figs. 9 and 10 suggests a great similarity between ore and wall rock. If the groundmass as shown in Fig. 9 were to be replaced by siderite, the two micrographs would be identical.

*Replacement after Major Folding of Rocks*

The main occurrence of the metadiorite rock trends in a north-south dircetion virtually at right angles to the general trend of the other formations and cuts across the West ore body. Also, it cuts across a portion of the banded silica member of the north side of the ore body.

At the south contact of the ore body, the metadiorite has in part cut across the banded silica member and has also included within itself relatively undisturbed portions of the banded silica member. In the sections included within the metadiorite, the rock between the silica bands has been replaced by chlorite, so that now the banded silica member consists of interbanded silica and chlorite.

It has been shown that the metadiorite has invaded and replaced the pyroclastic rocks subsequent to the development of a schistose structure within the pyroclastics. It has been noted that the shape and distribution of the metadiorite outcrops indicate an intrusive relationship with respect to the pyroclastic rocks. The structural evidence thus indicates that the metadiorite is not only younger than the pyroclastics and the banded silica member but that also it is apparent the metadiorite developed in its present position after the major folding of the pyroclastics.

It is believed that the siderite replaces metadiorite and thus is younger than the metadiorite. This opinion results from a detailed study of the siderite metadiorite contacts.

A determination of the contact between siderite and metadiorite is difficult because it is often gradational and of an irregular nature. It is usually a matter of estimating the amount of undigested chloritic material that has been included in the siderite but that has not been completely replaced.

Sometimes remnant structures that appear to be old dikes or offshoots of the metadiorite will be encountered in mining in which the replacement has been so complete that little concern will be given the quality of the ore other than to note a darker color and slightly coarser texture.

Sometimes, as may be seen in Fig. 11, stringers of iron carbonate are present in metadiorite and cut across a rather poorly developed schistosity in the metadiorite.

Often, in an examination of the walls of the open pit, siderite will be observed to truncate a schistose structure in the metadiorite and at the same time there is never any evidence that siderite itself has been subjected to the process that developed the schistosity in the metadiorite.

Additional evidence that siderite has been introduced after the major folding of the rocks is the fact that locally narrow bands of siderite have been observed within the pyroclastic rocks, outside the ore body, in which the siderite band truncates the schistose structure of the pyroclastics.

### *Siderite Introduced into Banded Silica Member*

The exact contact between the siderite ore bodies and the banded silica member along the north side is usually difficult to define. Throughout most of the contact, and certainly along the entire length of the north contact of the East ore body, there is a brecciated zone in which the banded silica material has been broken into innumerable fragments ranging in size from ¼ in. to 2 or 3 in. in diameter. The matrix of the brecciated material is siderite or siderite and pyrite. Locally, as along some parts of the north contact of the West ore body, brecciation is not so thoroughly developed, and in these places bands of quartz one inch wide are present within the ore body 10 ft. from the main banded silica wall. The quartz bands may be one foot or more apart. It is usually when the spacing between alternating bands of siderite and quartz is reduced to an inch or two that actual contact with

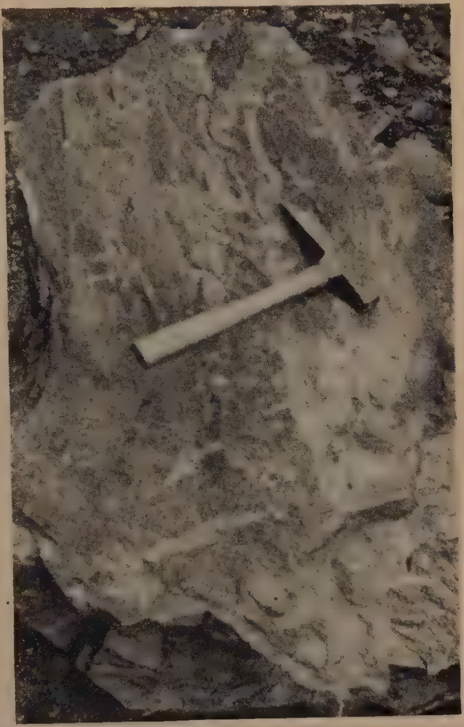

FIG. 11.—STRINGERS OF IRON CARBONATE WITHIN METADIORITE.
"Narrows area," West ore body, open pit.

the banded silica member is regarded as established.

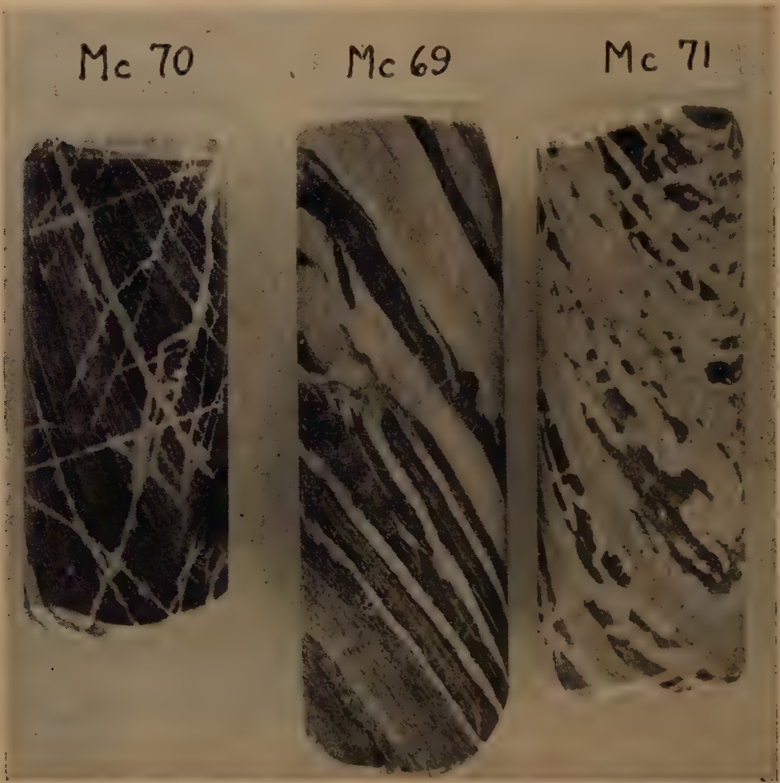

FIG. 12.—SUCCESSIVE STAGES IN INTRODUCTION AND REPLACEMENT OF BANDED SILICA BY SIDERITE. Specimens of drill core from D.D.H. No. 801; 415, 534 and 544 feet.

There is evidence throughout the banded silica member of the introduction of siderite along and between the bands of quartz, and often, too, examples of the siderite cutting across bands of quartz is seen. In addition to being introduced, the siderite also shows evidence of having attacked and partially replaced the quartz.

Fig. 12 illustrates successive stages in the introduction and replacement of banded silica by siderite. Specimen Mc-70 shows a few tiny stringers of light colored siderite developed along and between parallel bands in the dark quartz member; also, a number of tiny stringers, which definitely trend across the banded structure. In detail it may be noted that the siderite stringers are uneven in outline because of their tendency to unevenly etch their way into quartz by a replacement process. Specimen Mc-69 and Mc-71 illustrate more advanced stages in the process of the introduction and replacement, until, as noted in Mc-71 siderite, has almost completely replaced the banded silica.

Further evidence, and to the writers very convincing evidence, that siderite has been introduced into the banded silica member, and that it is capable of forming banded structures conformable with the older silica bands is given in Fig. 13. In this illustration the siderite that forms the entire lower half of the specimen is observed to have transgressed and replaced a portion of the banded silica and to have advanced by a replacement process along the banded silica member to produce a rock that in appearance is identical with many of the banded iron formations observed in the field today.

The outstanding feature that appears to have made possible the development of a banded silica and siderite rock is an important structural control whereby the siderite followed and replaced certain silica bands and followed and replaced tuffaceous rocks that were interbanded with silica. The field evidence suggests the later to have been the commoner process.

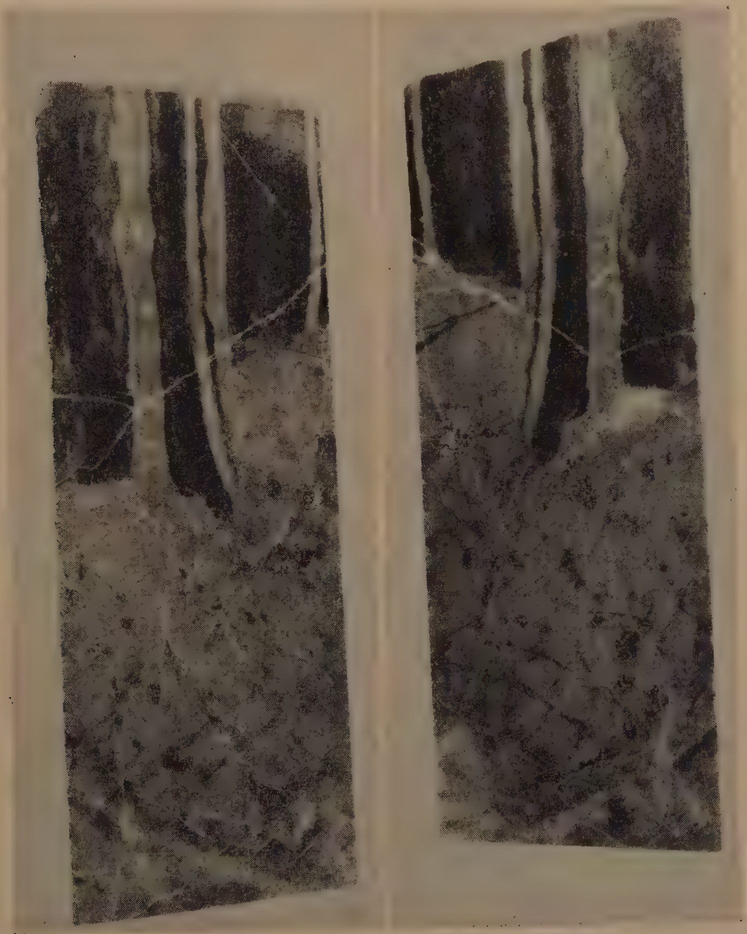

FIG. 13.—ILLUSTRATION OF PROCESS OF INTRODUCTION AND REPLACEMENT OF BANDED SILICA BY SIDERITE TO FORM BANDED IRON FORMATION.

Specimen of drill core from D.D.H. No. 801; 320 feet. Photograph by Geological Survey, Ottawa.

## ORIGIN AND DISTRIBUTION OF SULPHIDE MINERALS AND OF MAGNETITE

The only really important sulphide minerals present in the siderite are pyrite and pyrrhotite. These are definitely later than siderite and replace siderite. Other sulphide minerals are present; for instance, chalcopyrite, arsenopyrite and galena. All the sulphide minerals are relatively late in the mineral sequence and there is evidence of more than one age of pyrite.

The pyrite is most abundantly developed in the brecciated contact between the banded silica member and the siderite ore bodies. Usually it extends 25 ft. into the siderite but locally it may be developed abundantly across the north half of the siderite ore bodies. Replacement of siderite by pyrite may be relatively complete, so that little siderite remains. It is much more usual, however, for the pyrite to occur as clusters of crystals surrounded by siderite.

In the Old Helen mining operations, pyrite assumed considerable importance, for slightly more than 50,000 tons of pyrite was mined and shipped. Most of the pyrite mined in the Old Helen is described as pyritic sand.

Locally pyrrhotite seems to compete with pyrite in importance, and although it is usually present in abundance only within and adjacent to the contact of the siderite and banded silica, it has also been noted to extend well into and across the main part of the siderite ore. The pyrrhotite replaces the siderite, and replacement at times is sufficiently complete, especially at the contact of siderite and banded silica, to give rise to a banded pyrrhotite and silica rock instead of banded siderite and silica.

Chalcopyrite and arsenopyrite have been noted in the form of tiny seams, and the arsenopyrite has also been noted as individual crystals. They are usually seen in one contact or another of the ore body but never more than a trace of either mineral has been found.

Only one occurrence of galena has been noted and in this instance the galena seems to be associated with a relatively young ankerite stringer.

The pyrite and other sulphides are thought to represent relatively late products of emanation from an igneous source. The sulphides certainly are later than the siderite, which is also regarded by the writers to have an igneous source. Brown and Morrison[14] regard the pyrite as later than hematite. E. S. Moore[9] attributes the source of the pyrite in the Michipicoten area to Algoman intrusives.

Magnetite at times is an associate of siderite. At numerous places it appears concentrated adjacent to the contact with diabase dikes and at such places there appears to have been a reaction between the dike and siderite to form magnetite. In such instances actinolite is sometimes developed.

## Fold and Fault Structures

Of considerable importance in determining the shape of the siderite ore bodies are the dragfold structures that were developed in the rocks during mountain building prior to the introduction of the siderite. These structures are not only important in that they have exerted an influence on the shape of the ore bodies, but also apparently they have influenced the position of a set of cross faults, which developed at a much later time.

The faults are relatively young; i.e., they are younger than the siderite but are older than the diabase dikes. In the West ore body the cross faults are numerous (Fig. 14), and although not so well defined in the East ore body there is considerable evidence that they exist. The faults are essentially parallel structures, which tend to strike across the ore body nearly at right angles, but to the north and south of the ore have a tendency to curve and follow the general trend of the rocks (Fig. 5). The faults usually dip vertically or at very steep angles either west or east. Sometimes the fault plane is a rolling surface and may locally assume a low angle of dip.

There is usually an offset on each of the faults in such manner that the east side moves north relative to the west side. The amount of offset on the two contacts of the ore body along any one fault is practically never the same and usually one side will coincide with a relatively older fold structure. The offset is usually in the order of a few feet but in two instances at least it is in the order of 100 ft. The faults are approximately parallel to major regional faults mapped by Collins[4] and the relative offset is in the same direction.

The faults for the most part appear to represent planes on which there has been movement with mainly a vertical component. Local rolls in the contacts of the siderite, which cause the contacts to depart from a parallel position, give rise to an offset differing in amount on the two contacts. The direction of movement on the faults is east side downward relative to the west side. This means that as advance from west to east is made along the ore body, rocks that were at one time relatively higher in the earth's crust are encountered. Recognition of this structural condition within the ore body is of considerable importance because there is a distinct tendency for the silica content to increase toward the east. This is especially true when a comparison is made between the West and East ore bodies, because in this instance an increase in silica in the East ore body is in a large part due to the presence of a 50-ft. width of brecciated and banded silica, and this particular horizon is absent in the West ore body.

In the area between the West and East ore bodies in which two diabase dikes are present, there has been faulting of the kind described above. It is observed in Fig. 5 that the banded silica portion of the iron formation is only about one half as wide east of the dikes as it is to the west, and, also, it may be observed that the offset in the south contact of the iron formation amounts to 500 ft. and at the same time the north contact is not offset more than 100 ft. This feature strongly suggests that, in addition to some structural control from the original dragfolding, there has been later faulting and that at least two faults have been developed in the area of the dikes.

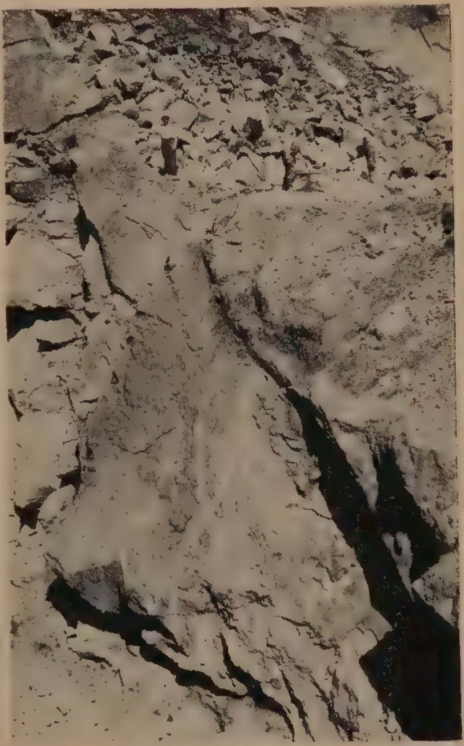

FIG. 14.—CROSS FAULT, NORTH CONTACT OF WEST ORE BODY.

Showing banded silica-siderite member on left, siderite ore on right.

## OLD HELEN ORE

Although this paper is mainly a discussion of the Helen siderite ore bodies, the geology of the Helen is not complete without mention of the fact that under Boyer Lake immediately west of the West ore body, there was mined a number of years ago some 2,823,369 tons of iron oxide.

This ore is usually referred to as "brown ore," and as far as known by the writers all those interested in the area, except possibly T. L. Tanton, attributed its origin to weathering of siderite to the oxide form.

In a previous part of this paper there are described a number of cross faults in the siderite ore bodies. It is shown in Fig. 5 that one of these cross faults is present in the area which, according to the old mining records, marked the eastern limit to which the Old Helen mining extended. Also in Fig. 5 is shown the eastern limit of the Old Helen underground workings on the bottom or the eighth level. It may be noted that the eastern end of the eighth-level workings is 600 ft. vertically below the exposed siderite on surface. The writers are of the opinion that more cross faults exist in the area to the west of those shown in Fig. 5 and that probably more than one of them were encountered in the Old Helen workings but were not recognized as more than local slips or seams.

The presence of cross faults no doubt is an aid to the process of changing siderite to the oxide form because channels are provided for surface weathering agents to follow. A number of cross faults have been examined during open-pit mining operations, to determine the extent to which surface weathering is influenced by them. A coating of brown oxide on the siderite has been noted for a depth of a few feet along the faults exposed but in no instance has the weathering extended below the first mining bench; i.e., more than 50 ft. below surface.

It is difficult to imagine conditions of weathering and oxidation that could extend to a depth of at least 600 ft. below surface in one area and immediately adjacent, within a few feet, excellent quality siderite present, and with the only evidence today of weathering of this siderite a film of brown ore on the surface, which is not more than a few inches thick.

Although most of the records that have to do with the Old Helen refer to the ore as brown ore, R. W. Seelye[3] mentions the fact that to the west of the Old Helen diabase dike (Fig. 2) a body of hard red hematite ore was mined on the third and fourth levels of the mine. Seelye reports the iron content of this hematite to be 60 per cent and the average for the mine to be 57 per cent.

In view of the interpretation developed in this paper in regard to the origin of the siderite, the whole story of the origin of the Old Helen ore must be reconsidered.

In a further study of the origin of the Old Helen ore, weight must be given the evidence recorded by Tanton,[13] Roberts and Bartley,[15] and Brown and Morrison,[14] in regard to the origin of the Steep Rock and Josephine ores.

Also, since faulting is known to be an important feature within the siderite, it undoubtedly plays a part in the structural relationship between siderite and hematite.

In the junior author's opinion, the origin of a great part of the Old Helen oxide ore may be attributed to the same type of agency that produced the siderite. The available evidence at the Helen, along with that recorded by Brown and Morrison[14] at the Josephine indicate that the oxide ore is later than the carbonate. There appears to have been local alteration of iron carbonate to iron oxide, and there appears to have been local introduction of material that is present today in the oxide form. The agencies that have brought about the development of oxide ore are considered to have an igneous source.

## SOURCE OF THE IRON AND ITS POSITION IN THE GEOLOGICAL TIME SCALE

It has been shown that the ores are the product of introduction of iron-bearing material accompanied by replacement of the rocks already present. It is shown that introduction took place subsequent to the development of the major folded structures. It is believed that the iron-bearing material emanated from an igneous source.

The area adjacent to the ore bodies contains abundant evidence of a period of introduction of basic igneous material; i.e., the metadiorites. Also present in the area and about one mile distant on surface both to the northeast and south are present outcrops of granite.

Since the siderite is intimately associated with the metadiorite, its source can be attributed to the igneous activity that gave rise to and followed the introduction of metadiorite. Since the granites contain ankerite stringers and veins, it is evident that the period of introduction of iron carbonates extended into and beyond the time of formation of granite. The granites are regarded by Collins[4] as Algoman in age, and since the iron carbonates are attributed to the igneous activity that culminated in the introduction of granite, the carbonatization may be regarded as Algoman in age. The oxide ores are of age similar to that of the siderite, or only slightly younger.

## ACKNOWLEDGMENTS

Thanks are due to the officers of the Algoma Steel Corporation, Limited, for permission to present this paper, and to the members of the staff of Algoma Ore Properties, Limited, who assisted in its preparation.

It has been the authors' privilege to discuss many of the points in regard to the Helen geology with Dr. E. S. Moore, of the Ontario Department of Mines, Dr. T. L. Tanton, of the Geological Survey of Canada, and Mr. W. F. Morrison, geologist for the Josephine mine. Thanks are given Dr. Tanton for the preparation of Fig. 13 and Mr. Don Davis, of the American Cyanamid Co., for the preparation of Figs. 8 to 10 and Fig. 12.

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# Geologic Relations and New Ore Bodies of the Republic District, Washington

By Lawrence B. Wright,* Member AIME

## Abstract

The Republic district, Washington, is of new interest because of the discovery of new gold-silver ore within a unique structural pattern. The camp was revived in 1937 following installation of a plant to treat low-grade disseminated ore at Knob Hill and further stimulation followed discovery of "cross vein" breccia ores in 1939.

Previous writers [1-2] ascribed the quartz veins to a latite porphyry stock, but are in disagreement as to its precise location and extent. This study leads to the conclusion that the ores have a more profound parentage linked to a broad system of thrust faults and folds, the mechanical origin of which is related to a southward trending (Sanpoil) syncline connecting with the Lewis and Clark zone of shift described by Billingsley and Locke[3] in 1939.

The Republic anticline and a cross anticline recumbent against Cretaceous granodiorite, and involving some 4000 ft of Eocene andesite flows and early Miocene lake beds, localize the ores, mainly along the east flank, in a series of openings simulating drag folds and steeper than the folded flows.

These broad structural features no doubt reach deeply into the basement rocks, where at appropriate crossings and junctions it is suggested that heat effects gave rise to ore-making materials, including, in this instance, intrusive breccias that later were mineralized, and comprising the broader bodies of new ores. The breccia ore is seen crosscutting the earlier narrow quartz veins with a welding of the two at junction areas by a second surge of ore minerals. The ore mined and developed since 1937 equals district production from 1896 to 1937.

The ores are typified by cherty quartz, adularia and some barite, with fine pyrite and marcasite, polybasite, pyrargyrite, tetrahedrite and gold. Selenium is present, possibly with gold, as a normal selenide.

The disseminated ores, as well as the vein and breccia ores, are localized principally in one 500-ft flow (Knob Hill andesite), which has been traced around the crest of the low northward-plunging Republic anticline. The gold-silver ratio in the earlier narrow veins is about 1 to 4 and in the disseminated and breccia ores, 1 to 7. The latter higher silver ratio is deemed to reflect the character of the second surge of mineralization.

Sills and dikes of the Columbia River (Miocene) basalt cut the folded structure and ore, thus placing the mineralization in the middle Miocene.

## Introduction

The Republic district of north central Washington is of new interest because of the discovery, development and extraction of new gold-silver ore; also, because of a unique local structural pattern believed to be integral with the broader tectonic mechanism of the continental framework.

This district had 40 years of modest rating following its discovery in 1896. Its narrow ore bodies, complex mineralogy, and early inadequate metallurgy, together with the apparent bottoming of ore near 1000 ft, limited production to 300,000 oz of gold and 1,000,000 oz silver between 1897 and 1914. The camp was then reduced to lease life until 1937, when the gold price increase of 1933 to $35 per ounce

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Manuscript received at the office of the Institute July 15, 1945. Issued as TP 2197 in Mining Technology, July 1947.

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[1] References are at the end of the paper.

stimulated renewed activity. This brought in a new organization that provided a cyanide plant of 500 tons daily capacity, for the purpose of treating disseminated low-grade ore. The ores mined and developed from 1937 to 1945 equal in value the total prior yield of the district.

Discovery of the new higher grade breccia ore bodies (South Cross Vein System) in the north end of the district occurred in 1939. The sequence of events leading to this were: (1) recognition of a cross-fault system, (2) observation of mineralized breccia in similar near-by fault zones, (3) encountering disintegrated subsoil quartz and breccia with increased values in the cross-fault area in the open pit, (4) drilling three churn-drill holes in the fault-zone hanging wall to test for ore in place down the dip, and (5) drifting to the ore from old mine workings.

To what degree is this a reproducible experiment? Does an unfolding of structural pattern and ore control facilitate discovery? Of what help were previously published conceptions that the known ore must be related to a visible stock and that therefore a stock must be present with veins conveniently arranged thereabout?

As a result of field observations before, during, and after the discoveries, these and some other questions can be answered in this paper. It will: (1) emphasize a new conception differing widely from published views, (2) show the relation of the district to the region, (3) show the place of the ore in the district, and (4) present new data relative to details of mineralization localized in a structural trap.

## CHANGE FROM PUBLISHED VIEWS

It has been the custom to postulate a stock below or within an ore area, and to delegate to it all of the mineralizing powers. In this district it is evident that the strength of that idea led the early observers to map outcrops approximately where a stock should be, show them as a stock, and charge it with parenthood of the ore.

Actually, these outcrops are composed of tilted, folded and faulted flows of andesite cut by dikes with the composition of diorite. Ore occurs at places in the entire suite. In this study the attempted application of field facts to the old orthodox scheme brought only inharmonious patterns and left no room for possible discovery of large bodies of ore. Certain features not included were: (1) the district anticline, (2) thrust faults against the granodiorite, (3) younger ore cutting older veins, (4) traps, (5) intersecting fault zones, (6) cross folding, (7) district basement influence on depth.

These seven important features of the over-all pattern force a new conception of the district. The more remote questions as to why the ores favor the east flank of the district anticline, and the relation of this structure to the larger continental framework, must be treated speculatively, as the study in detail has not gone far enough away from the ores.

## LOCATION OF DISTRICT IN REGION

According to Billingsley and Locke,[3] the Columbia Basin and Great Basin are Pacific blocks that moved southeast a known maximum of 120 miles, and, at the Wasatch Front broke into waves against the high continent platform. The southwest edge of this block is made by the Walker zone of shift. The northeast edge is made by the Lewis and Clark zone of shift.

The Lewis and Clark zone strikes west-northwest from the Yellowstone past the south end of Boulder batholith, to which it is tangent. Thence it echelons 60 miles to the north end of the batholith and proceeds west-northwest as before. The batholith that contains the Butte ores lies, it is believed, on a west-dipping thrust between two members. The Lewis and Clark zone strikes along the Osborn fault through the Coeur d'Alene district, thence along the north edge of the great Columbia lava field and toward the Pacific coast.

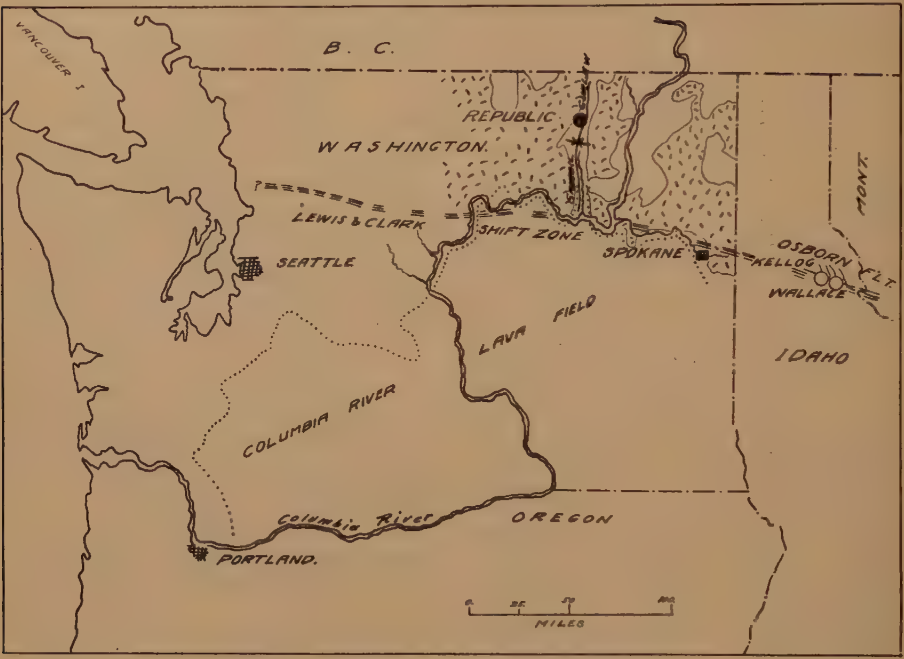

FIG 1—LOCATION OF REPUBLIC DISTRICT.

This zone, which is old and even now "alive," as the current Montana movements prove, has various crosswise structural neighbors in addition to the Boulder and Phillipsburg thrusts.

One of these emerges from cover near the mouth of Sanpoil River at the north edge of the lava field 125 miles beyond Coeur d'Alene. It is a synclinal zone with strong axial faulting and is flanked by granite and metamorphics with lavas and sediments between. It strikes north and carries the Republic district of gold-ore bodies.

Do the main members of these broad structures cut downward to the "mineralizer sources" which have long been part of the idea of ore origin? Do the crossings or junctions make "release points"? Or, are these crossings or junctions strain points at which heat effects, including ore, are locally generated? These considerations yield a glimpse of one approach that search for hidden ores will require.

## REPUBLIC DISTRICT

The Republic district is made up of several gold-silver deposits in an area 6 by 3 miles. It is near the north end of an anticlinal drag fold near the 50-mile point (from the south end) on an 80-mile-long synclinal structure. The rocks in the syncline are, in the main, crumpled Tertiary volcanics, mostly andesite, with several exposures of Paleozoic sediments brought to view by truncation of subsidiary anticlinal folds. Columbia River basalts lie unconformably over part of the area and are widely spread to the south.

The main zones or lines of shift, including this one reaching northward from the Lewis and Clark line to Republic, are believed to extend through the basement zone to mineralizer sources. Source points are herein assumed to be occasioned at crossings or junctions. Such a crossing is seen in the Republic district where a westward anticlinal axis crosses the northward axis of the drag fold termed the Republic (Mud Lake) anticline. The ores are spread in a pattern structurally integrated with this anticline whose form was determined by a rock-crumpling thrust against the competent Granite Mountain mass. Their emplacement evidently was facilitated by the cross-fold influence as the east-west axis coincides with major mineralization along the Republic anticline.

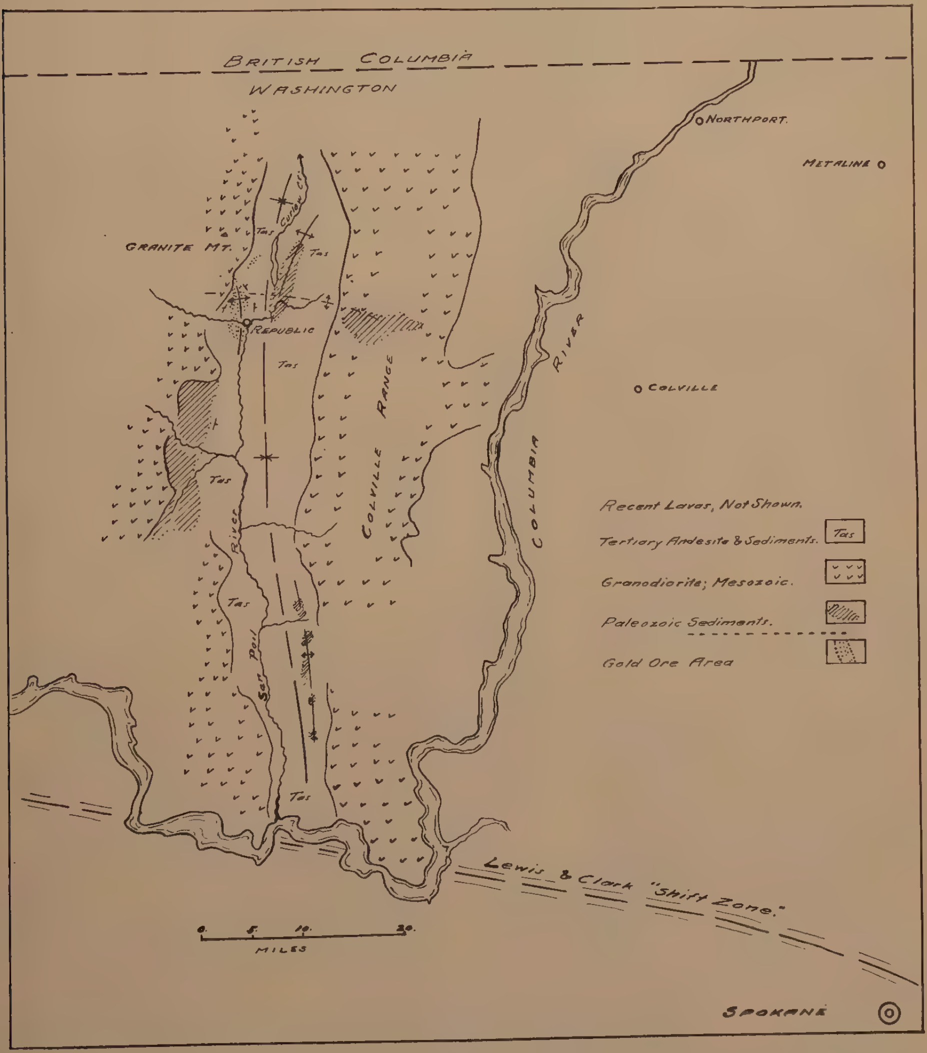

FIG 2—DRAINAGE PATTERN.

The Curlew-Sanpoil drainage pattern

is strikingly cut along the axis of the synclinal "channel" throughout its entire length. Curlew Creek flows north from a source about 3 miles northeast of Republic, while the Sanpoil River flows 50 miles south from the same source area. This division of drainage occurs 3 miles northeast of Republic and coincides with the cross anticline* axis (Fig 2).

## GENERAL GEOLOGY

The rocks of the district are described in detail by Umpleby[1] and by Lindgren and Bancroft.[2] However, their grouping and distribution as treated herein coincides with a different structural pattern.

Thickness of members is difficult to measure. Repetition by faulting and folding is common. Table 1 gives approximate thicknesses of members from transverse sections and surface observations.

TABLE 1—*Approximate Thickness of Members*

| MEMBER | THICKNESS, FT |
|---|---|
| Basalt (Columbia River) | 700 |
| Quartz diorite-dacite intrusives | |
| Andesite flows (upper) | 1,000 |
| Shale with sandy layers | |
| Andesitic sills included (Mud Lake series) | 900 |
| Andesite (Knob Hill series) | 525 |
| Andesite (lower, fragmental) | 1,500 |
| Granodiorite (Granite Mountain) | |
| Limestone, cherty shales and other basement rocks | 5,000 |

The ore deposits are arranged along the flanks of an anticline. The axis of the anticline trends northward through the vicinity of Mud Lake. The nose of the fold plunges flatly northward and is last seen north of the Mountain Lion Mine.

Most of the ore occurs along the east flank, emplaced in a series of openings having dips steeper than the enclosing rocks. A feldspar-rich series of flows, termed the Knob Hill andesite, is the principal host. These flows are exposed at the New Republic mine, at the south end, and northward the ores are exposed along Eureka Gulch and Knob Hill mine area to the nose of the fold. The west flank ore, except for the Mountain Lion mine, has been less important. The andesite host on this limb of the fold is recumbent at shallow depth on the granodiorite of Granite Mountain along the main thrust zone (Fig 3).

The Knob Hill andesite is overlain by Miocene shales grading upward into sandy beds with intercalated tuffaceous flows; all are participants in the folding. Beneath the Knob Hill andesite there are tuffaceous and coarsely fragmental flows resting on the folded Paleozoic basement.

Intrusive into the whole are several dikes of dacite and diorite. (The quartz-latite porphyry of J. B. Umpleby[1] and Lindgren and Bancroft[2]). Two of these dikes are seen cutting folded sediments ½ mile west of Republic. The Ben Hur vein in the Trade Dollar mine is in similar diorite. A dacite dike in the Mountain Lion is clearly pre-ore. Dike and ore are both cut by (Columbia River) basalt.

The Flag Mountain porphyry emphasized by both Umpleby and Lindgren has been regarded as intrusive and the ore parent, but recent study of thin sections shows it to be one of the more compact flows. It is folded and crumpled, as are the others. It has lent itself to severe attack by mineralizing solutions in the "spread" zone. It is therefore indicated on the general map and sections

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* Cross Anticline: details in support of the presence of a cross anticline are generally lacking in the accompanying illustrations. It is not readily discernible in the field, but is present and evidenced by: (1) the Sanpoil-Curlew drainage divide, (2) Paleozoic sediments brought to view east of the divide (3) openings paralleling the axis and some filled with dikes, (4) drag folds, in mine workings, whose axes parallel the cross axis, (5) reversal of dips on either side of the drainage divide, and (6) mineralization along the trend of the cross-fold axis.

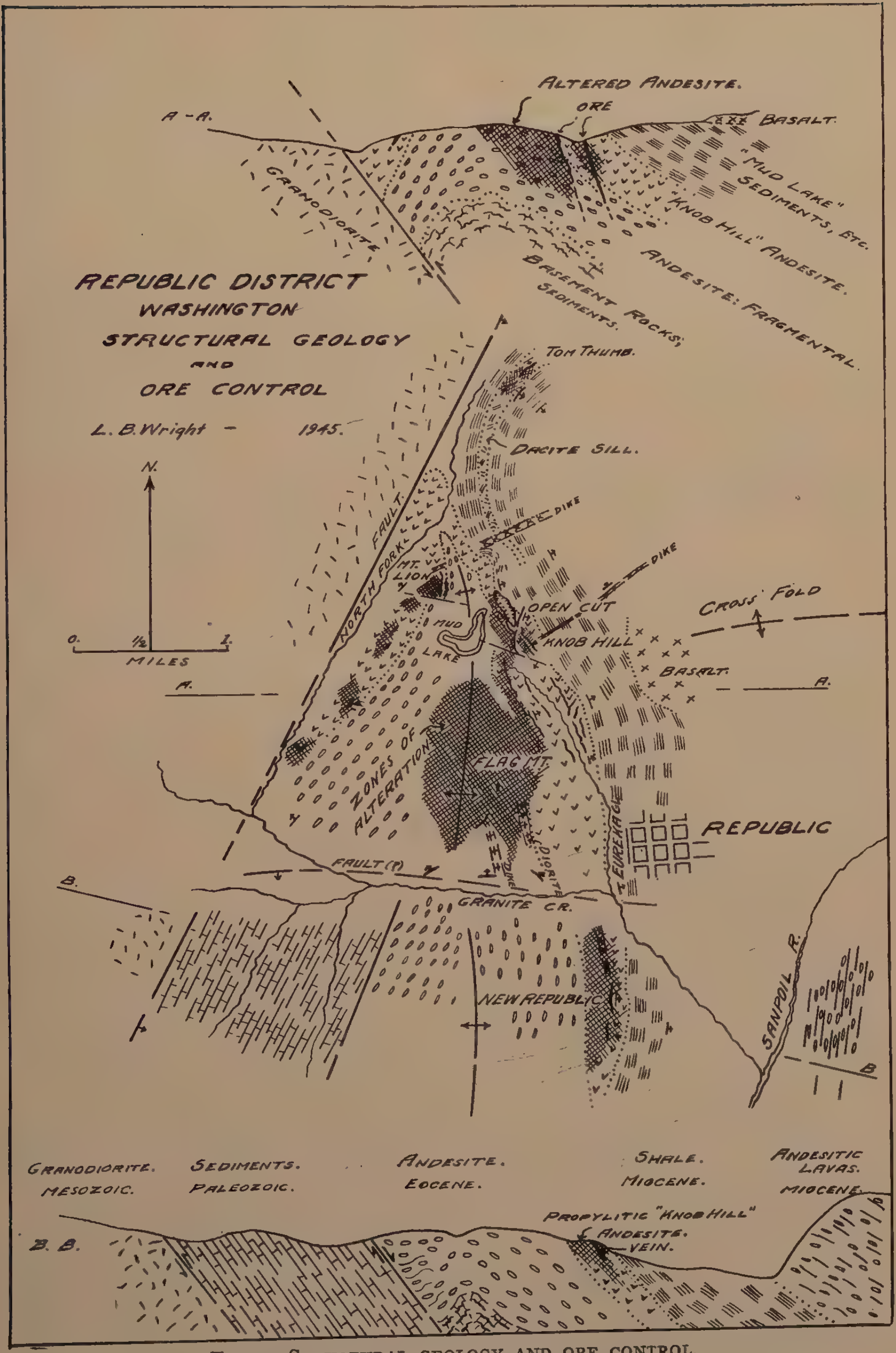

FIG 3—STRUCTURAL GEOLOGY AND ORE CONTROL.

as an altered mass and is shown in the legend similar to altered areas of the other andesites.

### Sequence of Geological Events

The sequence of geological events is tabulated as follows:

1. Deposition of Paleozoic and Mesozoic sediments.
2. Intrusion of Granite Mountain granodiorite (Cretaceous).
3. Erosion.
4. Extrusion of 4000 ft of andesitic flows progressively becoming more finely crystalline near the top (Eocene?).
5. Erosion. Deposition of shallow lake sediments (Miocene).
6. Thrusting, folding and early faulting in a pattern simulating drag folds.
7. Intrusion of dacite and diorite dikes and sills.
8. Ore deposition. Quartz, adularia and calcite gangue filling narrow fissures. Gold-silver mineralization associated with coarse pyrite as in the South Lode vein. Gold-silver ratio about 1 to 4.
9. Faulting along vein walls. Cross faulting brecciating veins at intersections.
10. Intrusion of dark cherty breccia "dikes," following faults along vein walls as well as cross faults and in some cases following andesite flow boundaries.
11. Mineralization throughout the area, in part superimposed on the older mineralization, as in the Mountain Lion and in the South Lode vein where it abuts the later South Cross vein breccia ore. This mineralization is typified by selenium, some sulpharsenides and finely crystalline pyrite. Marcasite probably near end or later. Gold-silver ratio about 1 to 7.
12. Post-mineral faulting. Example: the Center Fault, Knob Hill No 2 mine (Fig 4).
13. Basalt intrusion. Example: basalt sill cutting Mountain Lion ore without lateral displacement. (Late Miocene).
14. Erosion by glaciation; at least 1000 ft removed.
15. Moderate subsequent erosion.

### Faulting and Folding

Premineral faulting occurred during folding and most of these openings parallel the axial planes, hence the northwest trend of most of the veins. The rocks, having been under shallow cover, display a minimum of warping. The folding is not the type common to older and more deeply seated rocks where a higher degree of plasticity results in sharp folds with little faulting. The Republic anticlinal structure was accomplished by mild warping to the point of yield, followed by breaking and block movement. This is commoner with the andesites than with the overlying more elastic shales. Veins having dips steeper than stratigraphic boundaries, and striking parallel to structure lines, which often are somewhat divergent from stratigraphic trends, are observed. Such was the control influencing the pattern of the first mineralization.

Just north of Knob Hill the flows are shifted about 400 ft to the west along a pre-ore fault zone trending southwestward and having a subsoil outcrop. There are also numerous instances of horizontal ore displacement in the order of 2 to 60 ft in both the Mountain Lion and Knob Hill workings. Postmineral faulting is thus clearly evidenced.

However, the pattern as now developed is even more complex. In the interval between premineral and postmineral faulting, the early veins were faulted and brecciated. The intrusion of dark, cherty breccia "dikes" along fault planes and elsewhere was followed by additional more important mineralization. Fig 5 shows this relationship and sequence. The South Lode vein is seen to abut a mineralized breccia zone of much better ore, the new ore now known as the South Cross vein.

This sequence was obscured by "welding" of the new vein to the old by the second surge of ore and gangue minerals in the area of juncture. That a second mineralization enters the scheme is evidenced by the enrichment of the South Lode vein beneath the trap formed by the juncture, while the breccia vein above the juncture is leaner. There is no change in values in the postmineral faulted segment of the breccia ore (South Cross Veins) where offset 60 ft by the postmineral Center Fault (Fig 5).

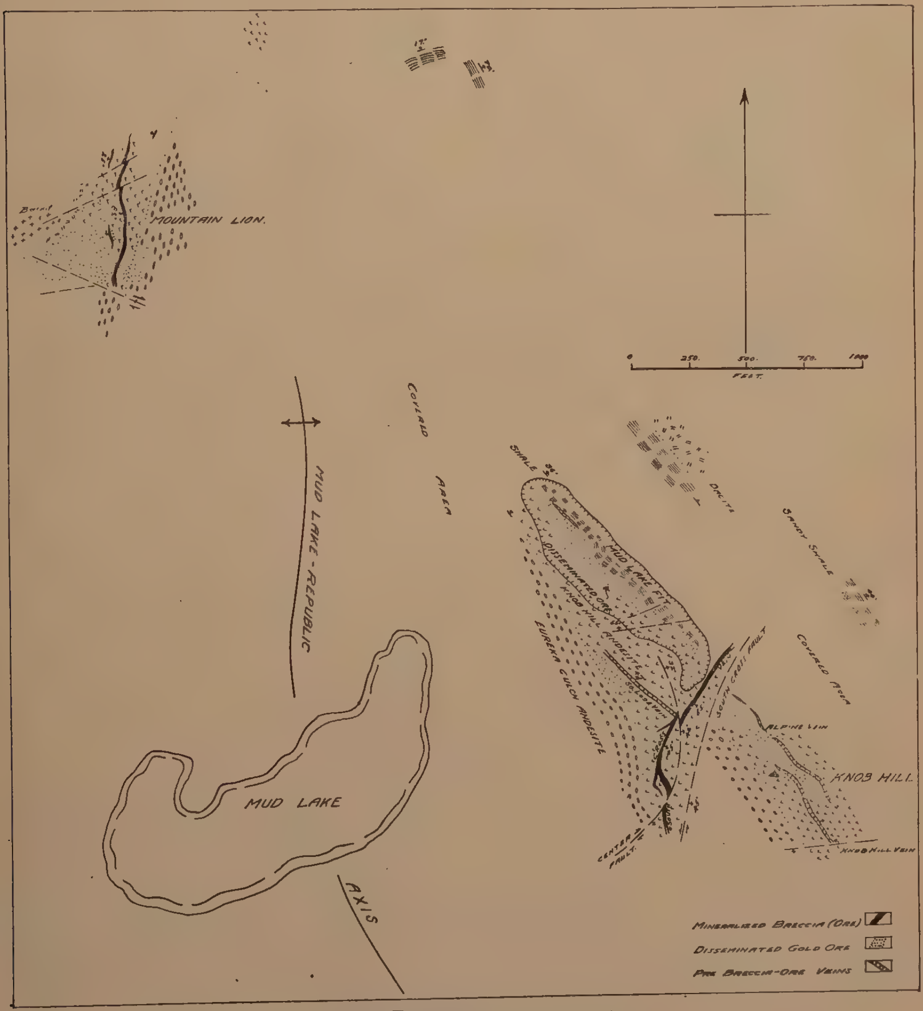

FIG 4a—POSTMINERAL FAULTING.

The early operators passed through the breccia vein in a low-grade section *outside the trap* and encountered the South Lode vein near the No. 2 shaft where it was too poor to be profitable. They were within 30 ft of ore worth several million dollars. Because they were conditioned by earlier conceptions to expect ore in quartz-filled fissures, they also missed the Stewart ore body by about 6 ft, farther to the north in the same working.

## VEINS AND ORE BODIES

At the Knob Hill mines there are two types of ore veins and two distinct types of disseminated ore. The Knob Hill and

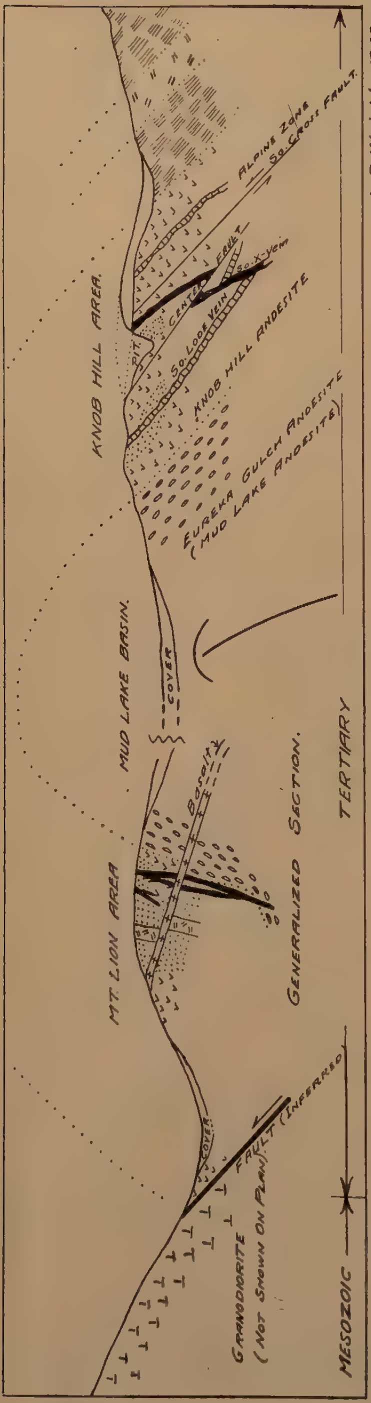

FIG 46—FOLDING AND POSTMINERAL FAULTING.

Alpine veins which supported the pre-1914 operations are typical fissure-type veins (4 to 5 ft thick). They are among the early discoveries. Workings were bottomed on the 12th level, the mine having been carried to this depth on the old Knob Hill vein.

The South Lode vein, also explored by the old operators, is similar in mineralogical composition but with greater average width. The quartz is somewhat more banded and spread over greater widths along a zone of shearing. The ore is diluted by unmineralized bands, fragments, and lenses of andesite. Oxidation, following planes of shearing, descends deeper in this vein than in the others, including the more recently discovered breccia ores in the Stewart and South Cross veins. Seams of limonite and bunches of manganiferous limonite are common, derived in part from coarse pyrite.

The ore zone of the new South Cross vein is unlike other ores of the district. The mineralized breccia is essentially flinty quartz and adularia replacing the "mud" matrix and surrounding angular fragments of dark chert. Perhaps it should not be called a vein, although within the mass the quartz shows bands and veinlike aspects where shearing of the breccia encouraged continuity. Frequently, nearly barren breccia occurs on either wall or within the ore zone. Stoping involves square-set timbering across horizontal distances from hanging wall to footwall up to 50 ft. The average stoping width is close to 22 ft. Fault gouge characterizes the hanging-wall contact with andesite, while the footwall contact is irregular but usually abrupt. This partly developed ore has been stoped for a length of 500 ft, and to 400 ft deep. The average grade is about 0.55 oz gold and nearly 4.0 oz silver per ton. The size, character and trend of these masses, not everywhere mineralized, suggest late intrusion of the breccia along zones of

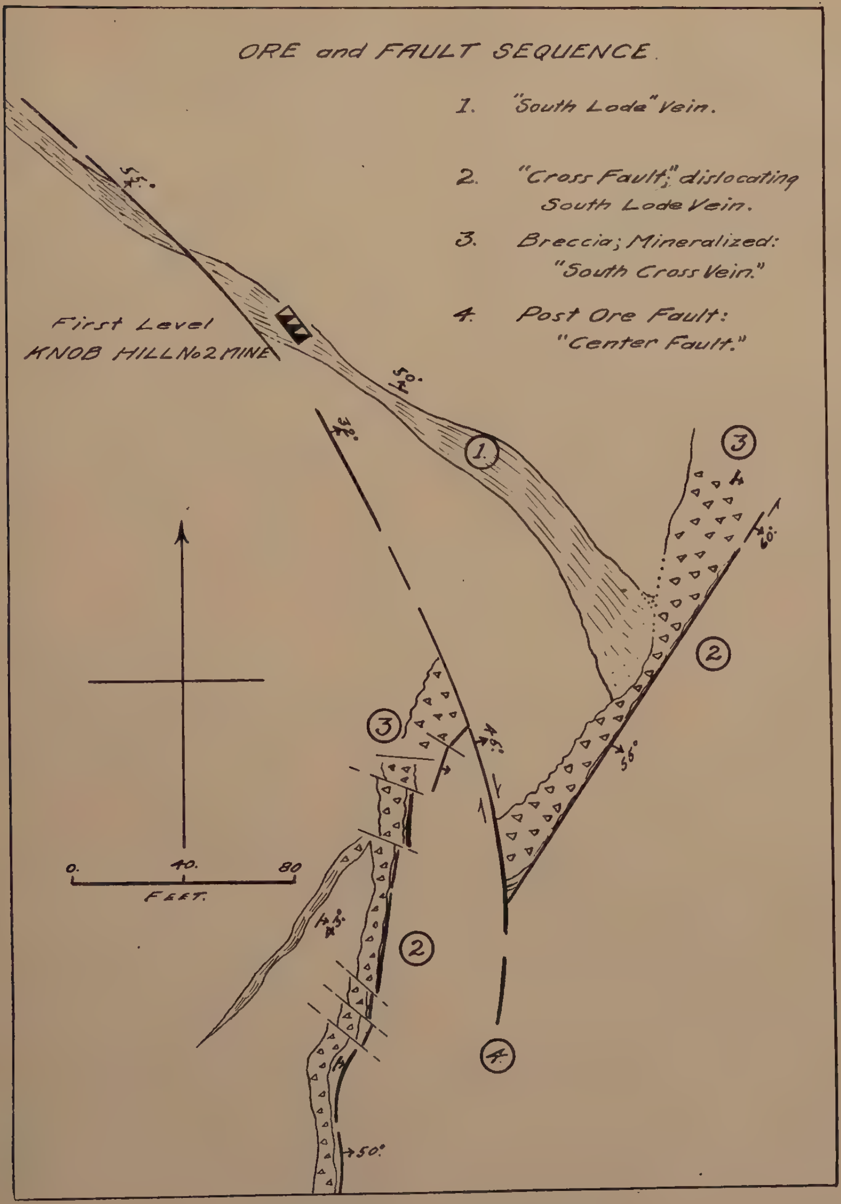
ORE and FAULT SEQUENCE.
1. "South Lode" Vein.
2. "Cross Fault;" dislocating South Lode Vein.
3. Breccia; Mineralized: "South Cross Vein."
4. Post Ore Fault: "Center Fault."
First Level
KNOB HILL No 2 MINE
55°
50°
38°
60°
45°
56°
45°
50°
1
2
3
4
0.
40.
80
FEET.

FIG 5—ORE AND FAULT SEQUENCE.

weakness generally discordant with the andesite flow banding. Fragments closely resembling the deeply underlying Paleozoic rocks have been observed in the breccia. This new ore is unlike the breccia ore described and illustrated by Umpleby,[1] whose observations related to quartz veins brecciated by subsequent faulting.

The Mud Lake low-grade ore is broadly disseminated in the upper (Knob Hill) andesite and reaches a thickness of 80 ft. It lies immediately below the carbonaceous clay shales, which, together with the glacial debris overburden, determined the economic depths of open-pit mining (1937 to 1939). These ores averaged about 0.11 oz gold per ton. The pit has a length of 1400 ft.

The Stewart ore body is composed of a stockwork of small quartz veins in a fine-grained pipe of medium composition, emplaced in the footwall of the disseminated ore. This ore is of special interest in that the finely crystalline host is later than the Knob Hill andesite and closely resembles the diorite porphyry enclosing the Ben Hur vein in the Trade Dollar workings. It is unlike the other ores in that the dense, hard host yielded reluctantly to the preparation forces. The openings were in a pattern only roughly harmonious with surrounding structure. The distribution of quartz in the 300 by 20-ft stockwork is irregular and in small veinlets. Fortunately their metal content is sufficient so that the mass is ore.

The disseminated ore at the Mountain Lion mine is similar in most respects to the Mud Lake pit ore. The average grade is about the same although pyrite is more abundant. The body is about 1000 ft in diameter.

### Comment on Early Conceptions

The north part of the district at least is considerably more than "a series of parallel fractures dipping from 38 to 80 degrees . . . opened in the volcanic rocks and . . . occupied by sharply defined veins averaging 3 or 4 feet in width."[4] This district has been regarded as being composed of narrow veins indiscriminately emplaced in an unsolvable complex of igneous rocks, the whole lacking definite pattern or structural control. Current mining widths reaching to 50 ft do not fit this early conception. Considerable importance is now attached to the extent and pattern of host-rock preparation by premineral events such as movements along regional planes, thrusts, folds, generation of ore materials in the source region, silicification, and intrusion of breccia masses.

### Mineralogy

The ore minerals are unusually fine grained. They occur most characteristically in concentric banding of cherty quartz, calcite and adularia.

From thin section and polished surface tests, pyrite, marcasite, pyrargyrite, tetrahedrite, polybasite, realgar and gold were identified in the South Cross vein ore, Second Level, Knob Hill No. 2 mine. The free gold is associated with the pyrargyrite and both are subsequent to tetrahedrite.

Selenium occurs in the ore and has been reported in other ores of the district. Lindgren and Bancroft[2] suggested the possibility of a normal gold selenide ($Au_2Se_3$) and pointed out that experiments have shown alkaline sulphide solutions incapable of changing the $Au_2Se_3$ ratio although attacking the free gold.

Free gold is seldom seen even in the richest concentrations, although occasionally its presence is denoted by a dark, bronzy, lusterless film and occasionally by straw-colored crusts. The characteristic dark bands, veinlets, and orbs, made up of the whole suite of gangue and associated metallic minerals, usually denote ore of average gold and silver content.

Swarms of minute pyrite suggesting replacement of organic forms indicate that some of the material in the breccia has been transported from underlying Paleozoic sediments.

The presence of selenium in the breccia ore of the South Cross vein places the introduction of this element and resultant minerals with the later surge of mineralization. More study is required before a dependable paragenetic diagram is possible.

Quartz, with which is associated the ore minerals, is seen replacing older quartz and carbonates having distinct veining.

The minerals so far recognized are: quartz, chalcedony, adularia, sercite, calcite, graphite, barite, gold, pyrite, realgar, marcasite, tetrahedrite, polybasite, pyrargyrite, and stibnite.

## GOLD-SILVER RATIO

A study of gold-silver ratios in the earlier narrow veins as compared with those in the broader ore masses shows that the latter were enriched to a greater extent by the second surge of mineralization. It also shows that silver had increased prominence.

TABLE 2—*Gold-silver Ratios*

| VEIN ORE | AU-AG RATIO |
|---|---|
| Northern part of district: | |
| Old Knob Hill vein, 7, 142 tons, 1911 | |
| Recovery adjusted to ore in place | 1–4.2 |
| Southern part of district: | |
| New Republic mine, 116 samples | |
| Reported by Umpleby, average | 1–4.1 |
| DISSEMINATED AND BRECCIA ORE | |
| Mud Lake pit, One month, 1939 | 1–6.6 |
| Knob Hill No. 2 mine, Year 1944 | 1–7.0 |
| Mountain Lion mine, One month, 1939 | 1–7.5 |

| Samples[a] | Au, Oz per Ton | Ag, Oz per Ton | |
|---|---|---|---|
| Specimen, 2 level South Cross vein | 1.8 | 40.0 | 1–22.2 |
| Six channel samples, 4 level, South Cross vein | 2.6 | 23.4 | 1– 8.9 |

[a] In local rich concentrations the ratio is erratic, but with usually an increase in silver.

These and other similar results are interpreted to verify the observation that gold and pyrargyrite were superposed on argentiferous tetrahedrite, and that crusts and veinlets of the latter probably served in part as a precipitant resulting in concentrations of silver-rich material.

## DISTRIBUTION OF GOLD AND SILVER WITHIN VEINS

The distribution of gold and silver along the trend is influenced by gentle warping of the zone. Mineralization is more intense in and near areas of changing strike and less intense in the long straight stretches. The slightly arched areas were favored because solutions were trapped in zones of increased permeability. The moderately synclinal areas, while mineralized, are less favored. The whole South Cross vein has the overall influence of the hanging-wall selvage, which functioned as a major solution trap.

Some low-grade ore is found in the hanging wall where small cross fractures permitted leaks in the structural roof. As development progresses, care is exercised to avoid overlooking some large leak leading to better ore in the hanging wall. No profitable bodies in this situation have yet been found. However, the Big Bonanza of the Comstock Lode was in such a hanging-wall appendage. High-grade ore in smaller spurs was discovered in the hanging wall at Ymir, B. C.[5] and at Neihart, Mont.

Spur veins angling into the footwall have been the source of some rich ore. In the No. 1 and No. 2 spur veins the walls are hard and abrupt. The gangue is in veinlike arrangement and the amount of gold and silver per unit of vein in the No. 1 spur is greater than in the main ore zones. The angle of departure of the spurs from the main body is 47° in one instance and 18° in another. Small footwall spurs* with a 90° divergence extend only a few feet.

At distances of about 130 ft from the junction, values in the two principal spur veins diminish to subcommercial grade. Within the next 20 to 30 ft the veins become indistinct and die out in hard, fragmental andesite of the deeper flows.

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* Where spurs extend only a few feet and can be mined within the main stoping pattern, they are regarded as a part of the main vein, but where they extend beyond 20 ft within their own walls, constitute economically separate units, and require a stoping pattern suited to their particular dimensions, they are given distinctive names and treated as separate ore units.

### STRUCTURAL CONTROL AND RAKE OF SHOOTS

Most of the ore on the east flank of the district arch is localized along 55° to 65° thrust faults paralleling on strike the moderately tilted andesites and shales and slanting across them down the dip into deeper flows. Broadly, two parallel "breaks" about 700 ft apart localize the ore along the central part of the district. Between these are important ore bodies in connecting diagonal openings. That the main thrust was upward toward the northwest against the granodiorite is evidenced by strong striae rising on a rake of 55° to the northwest in the fault selvage. Fragments of wall rock embedded in the clay gouge are rounded and each has left a 10 to 20-in. trail pointing downward to the southeast.

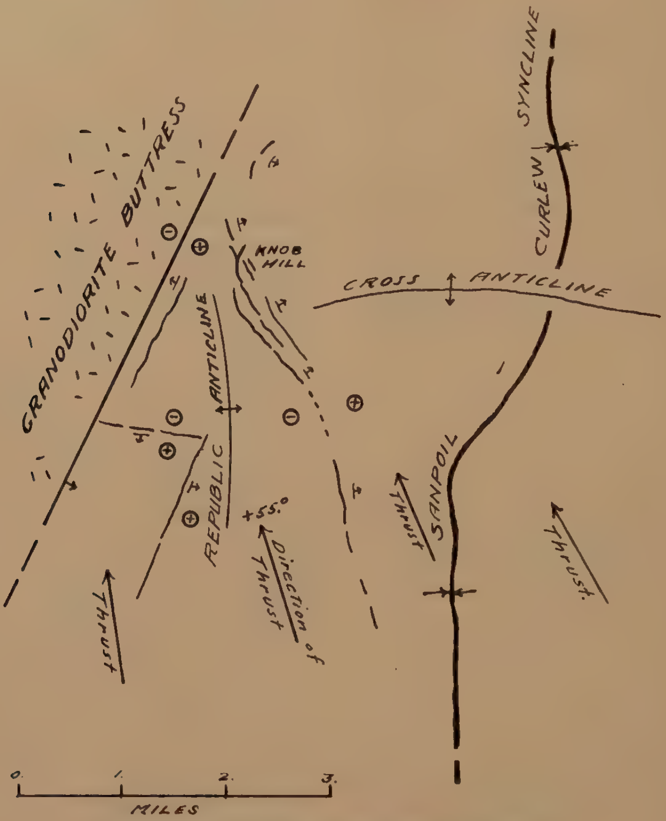

FIG 6—PLAN OF FOLDS AND FAULT PATTERN.

Throughout the district ore was localized along members of a fault mosaic and the direction of thrust determined the rake. The breccia ores are also localized along fault planes cutting the vein-type ore.

The disseminated ores are localized in silicified, slightly fractured andesite. Spread of solutions resulting in these larger lower grade bodies was facilitated by the low-dipping flow bands tapping the steeper main thrust planes; smaller expressions of the controlling district Y.

Within the zone of deposition the major openings that facilitated the spread of mineralization are members of the thrust pattern and the minor openings appear to be associated with the details of folding. The section (Fig 7) illustrates the factors favoring the stronger distribution of ore in the steep-dipping openings along the east limb of the fold. The lack of substantial mineralization along the anticlinal axis and along the west limb is attributed to less adequate conduits. This is particularly noteworthy because the axial fractures would normally be expected to be important channels. In this instance only weak veins (such as the Iron Mask) are seen in the axial zone. The region of source is believed to be a deep crossroad of geochemical activity from which, under proper auspices, ore-making materials escaped through the mobile zone to the zone of deposition. The zone of deposition in this district commences at about 1500 ft above sea level in the ore channels so far explored. It may be at a different horizon in the adjacent channels.

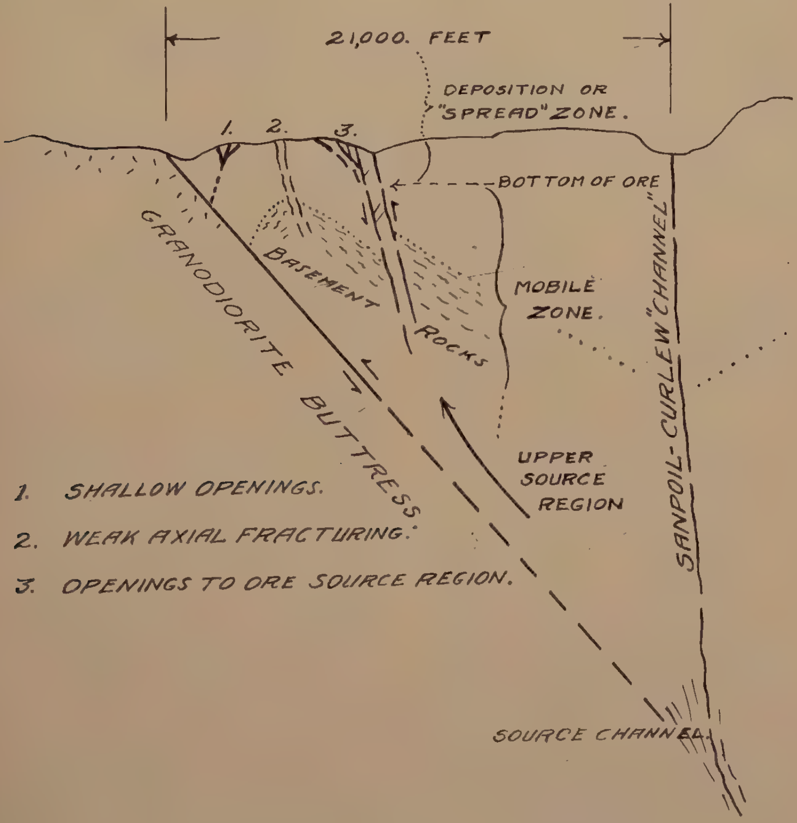

Fig 7—Schematic diagram of ore control.

Referring again to Fig 7, it is inferred that the No. 3 system of channels,* containing the bulk of ore so far known, detoured the solutions from the source region away from channels 1 and 2. Also, we may infer the approximate depth of the source region as being about 20,000 ft, or about 19,000 ft below the bottom of the old mines.

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* In Fig 7 the Sanpoil-Curlew synclinal axial plane is designated a "channel" and is so treated in dealing with the source and travel of ore-making materials. It is shown as being nearly vertical, and this might be questioned on the grounds that this "channel" has not been observed, and that major thrusts dip westward along the Rocky Mountains to the east. However, this axis is one of several in a series of northward-trending symmetrical

In this conception, the Republic ores (and possibly some dacite and diorite dikes preceding them) result from activity in a small segment of a geomechanical structure related to a broad regional structural pattern.

### The Ore Trap at Knob Hill

A structure resembling a segment of a cone, created by a series of fault zones, some of which were filled with early quartz and others with later intrusive breccia, formed the structural trap within which the most important ore occurs. The distribution of ore suggests channeling of solutions by the openings related to and a part of the trap structure. Some degree of "soaking" within the trap is shown by a consistent lean-gold distribution in footwalls.

The diagram of ore distribution and control (Fig 8) shows the pattern of this structure whose east member, composed mainly of cherty breccia, offered the most hospitable deposition loci. The dotted areas represent intensity of mineralization. The northeast member is the South Lode vein, which was early in the ore sequence. It was low grade initially, and being impermeable, was only moderately enriched. The north member is an east-west fault probably related to the cross folding. Beneath this fault, solutions have enriched the dense, gray brittle Stewart intrusive.

Thus, in a district localized along the flank of an anticline, the major ore occurrence is further localized by a very special complex structure or trap. Disseminated ore and narrow veins outside the trap area are important, but to a much lesser degree than the new ores within.

### Causes of Ore Termination

Ore in this district has terminated on strike or dip under observed circumstances tabulated as follows:

Case 1. Against postmineral faults.

Case 2. Against postmineral basalt.

Case 3. Against premineral dacite.

Case 4. In more coarsely fragmental flows, but not abruptly.

Case 5. As the vein approaches a strongly faulted zone, is brecciated and comingled with wall rocks, the per ton value is diluted below profitable grade.

Case 6. Temporary terminations occur along strike or dip where the vein continues but weakly through resistant rock, to emerge with greater width and value in a more hospitable host beyond.

Case 7. At the limits of premineral faulting.

Case 8. Temporary termination against mild (weak) cross fractures beyond which a continuation, moderately offset, usually is found.

### Depth of Deposits

A plotting of the bottoms of mines from the New Republic to the old Knob Hill, shows that mining ceased along a horizon near to 2000 ft above sea level. The Quilp workings reach to 1800 ft above sea level. This quite uniform lower limit of mining along a 6 mile zone where outcrops vary in elevation is too striking to be ascribed to other than some geologic reason.

In the north portion of the district, and perhaps throughout, the solutions

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folds whose crests are seen in traversing 150 miles eastward from the Okonogan Valley. These are in a block lying between the Cascade (Washington) and the Cabinet (Idaho) ranges. The Sanpoil-Curlew axis is near the middle of the block and, being in such a situation, the axial openings are deemed to be nearly vertical. The local thrusting and folding (Republic anticline) is a minor feature of the whole mechanism.

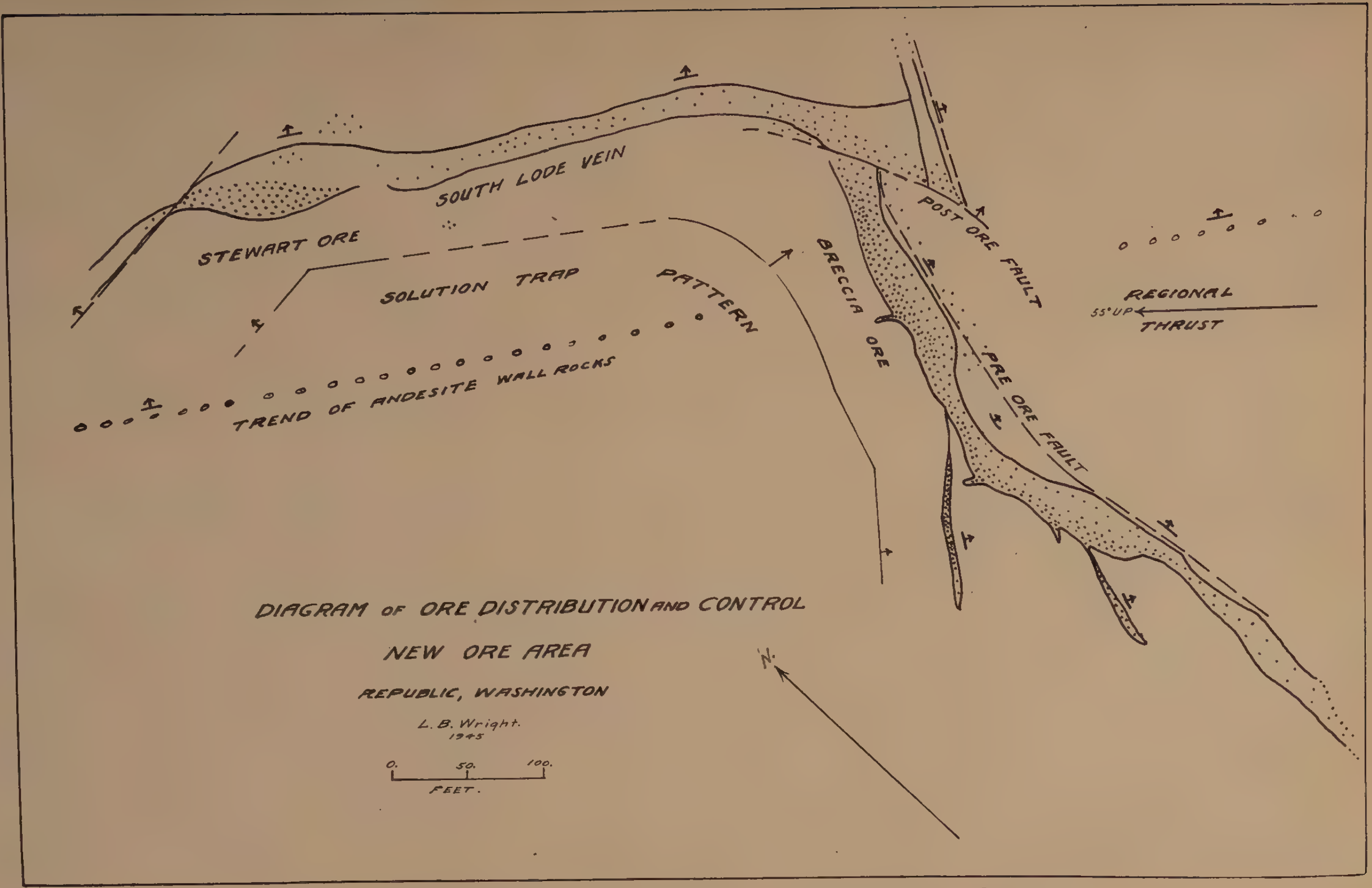

FIG 8—ORE DISTRIBUTION AND CONTROL, NEW ORE AREA, REPUBLIC, WASHINGTON.

have passed upward successively through basement rocks, metamorphosed Paleozoic sediments, coarsely fragmental andesites with alternating finer bands, and into 500 ft or so of medium textured holocrystalline andesite, the most favorable host.

The sequence is similar in the south end of the district, except that a greater thickness of Paleozoic sediments is indicated. The low northward plunge and gentle crest of the Republic anticline indicates that a common structural bottom may exist throughout. The crest of the folded Paleozoic sediments would, in such a structure, be irregularly near 1500 ft above sea level.

The present lower level of the new Knob Hill No. 2 mine is more than 1000 ft above this theoretical lower limit of ore.

Table 3—*Depth of Deposits*[a]

| Mine | Outcrop, Elevation, Ft | Workings, Depth, Ft | Bottom, Elevation, Ft |
|---|---|---|---|
| Knob Hill No. 1. | 3200 | 1200 | 2000 |
| Surprise........ | 2750 | 500 | 2250 |
| Quilp.......... | 2750 | 950 | 1800 |
| New Republic.. | 2700 | 700 | 2000 |

[a] Approximate.

These observations apply to ores along the Republic anticline, particularly the east flank. Obviously the structural bottom thus indicated would not determine lower limits of any ores localized along channels cutting favorable beds at horizons deeper than 1500 ft elevation, such as the Sanpoil-Curlew zone and the main granodiorite-thrust contact. Furthermore, there is nothing to rule out the possibility of a recurrence of ore in the Paleozoic sediments within the "mobile zone" (Fig 7). The "mobile zone" for one surge of mineralization could be the "deposition zone" for another in the same structure. The auspices under which they were launched determines the nature and extent of their journey.

## Details of New Discoveries

The Stewart and South Cross vein are the two ore bodies discovered during current operations.

The Stewart is west of and in the footwall of the Mud Lake pit disseminated ore. Its outcrop was uncovered in the course of normal open-pit operation, which required that soil overburden be removed (by bulldozer) at certain places along the footwall. Geology can claim no credit.

The South Cross vein discovery followed a different pattern. The sequence of events leading to the knowledge of the existence of ore in place in the breccia "dikes" along the South Cross fault zone were:

1. The cross fault zone and breccia dikes on the old Adit level were mapped in the first geological mapping in May 1939.

2. The Cove mine, south of and adjacent to the Knob Hill mine, was briefly examined in 1939. The breccia was seen emplaced along a strong fault trending toward the South Cross fault zone, swinging around the north side of Knob Hill, and displacing the andesites about 400 ft.

3. The Mud Lake pit operation had progressed to the area overlying the subsoil outcrop of the South Cross fault zone. The crushed rock, largely quartz, from the junction with the disseminated open-pit ore, the South Lode zone and the weathered talus from the breccia outcrops, led to deeper pit operation and better values in the area beneath the above mentioned "trap."

4. The suspicion that the better values in this "talus ore," as it was termed at the time, were due to the approach of the pit shovel to ore in place in a divergent trend, hinged on the conception of intrusive breccia along fault zones, and elsewhere.

5. Churn-drill holes 19, 20 and 21 were

laid out to test the more consolidated area downdip to the southeast, along the South Cross fault zone. The results were gratifying and the drive on the old Adit level (Knob Hill No. 2 mine, 1st level) then led into the new ore in place.

The degree to which this is a reproducible experiment is primarily determined by the possible existence of ore traps. A repetition of the series of events that led to discovery would assume a different pattern to the extent that direct geological attack now has the advantage of proved structural pattern. Adequate continued field study followed by drilling in critical places could lead to a repetition.

## SOURCE OF ORE

J. B. Umpleby[1] says:

> The open fissures in which the vein matter was deposited were probably all formed at approximately the same time. Their arrangement and distribution relative to the latite porphyry intrusion, suggests that the fissures resulted from stresses set up by activity in that body . . . latite porphyry outcrops at various places along the ore zone. Its largest exposure is along the east side of Flag Hill. . . .

Lindgren and Bancroft say:[2]

> Intruded . . . is a porphyritic rock which has been determined as quartz latite porphyry. Typical exposures of this rock occur on Flag Hill and Republic Hill . . . the maximum width of this intrusive stock is exposed just north of Granite Creek, where it occupies an area 1¼ miles wide" . . . the ore deposits are believed to be genetically connected with the intrusion of this porphyry.

Let us examine these observations and conclusions in the light of what may be seen in the field.

1. The two descriptions are at variance as to the location and extent of the "plug."

2. The "open fissures" could not all have been formed at the same time. There is later breccia ore emplaced in fault planes displacing the earlier quartz vein ore as much as 400 ft. Both ore types are dislocated 60 ft or more by postmineral faults.

3. The fissures are arranged and distributed not in relation to the "intrusion" but to a broader pattern of movement. That some of the ore lies in the alleged parent intrusive therefore no longer need be a mystery.

4. The cementing of quartz breccia by "primary" ore is, rather than being a "minor episode," one of the important events giving rise to better ore than has been known locally heretofore.

5. The formation of an ore district must meet the test of sound mechanics. Whether or not solutions are expelled from the magma after solidification of the outer part, as postulated by Umpleby, an ore district of this type and 6 miles long in all reasonable probability originated at least 4 to 5 miles deep. This is evidenced by the governing structural pattern.

6. Lindgren and Bancroft say:[2]

> Viewed in thin section the rock (quartz latite porphyry) is seen to have a holocrystalline ground mass composed of quartz and orthoclase. The phenocrysts are andesine, a very highly altered hornblende, biotite, and quartz. The hornblende and biotite are almost entirely resorbed. The rock may have contained augite. Apatite occurs as an accessory mineral. The hornblende is altered to calcite and chlorite, and the augite to calcite. These rocks may be difficult to distinguish from propylitic andesite, which shows weathered feldspar crystals that give the appearance of a porphyry.

From this view through the microscope they arrive at the following conclusion:

> The ore deposits are believed to be genetically connected with the intrusion of this porphyry.

This conclusion, based on thin sections of a silicified portion of the andesite series, is nonsequitur and unconvincing.

If there is any genetic relation it is because the array of events effecting ore emplacement, among other things, did not neglect alteration of the adjacent rocks. All of the andesites are to some degree propylitized and silicified. It does not follow that they are the ore parent.

A comparison of the Umpleby and Lindgren-Bancroft geological maps shows clearly the difficulty encountered in the attempt to relate the ores directly to a "quartz latite porphyry" plug. In fact, the maps disagree on the distribution of this rock to such an extent that doubt is at once thrown on the whole idea. For example, Umpleby shows the New Republic vein outcropping in the Lake Bed series near a small mass (0.9 by 0.12 mile) of latite porphyry, while the others show it in andesite near a mass of quartz latite over 2 miles long and nearly 1 mile wide (south of Granite Creek).

At the extreme north end the score is evened, but the area of disagreement broadened. Here Umpleby shows the Tom Thumb ore in andesite near a mile-long band of latite while Lindgren-Bancroft place it in the Lake Bed series with latite lacking.

These differences are important in that they indicate the lack of definite criteria supporting these earlier conceptions. This incongruity is especially emphasized when occurring, as it does, between such skillful workers.

A preferable view would seem to require recognition, first, of the broad structural features reaching beyond the district, resulting locally in channels serving as taproots and, second, recognition of the fact that on a continental scale mineralizing solutions in the Tertiary period sought release into prepared ground. Often they were preceded and sometimes followed, or both, by barren magmas. The genetic source in this case at least is ascribed not to the brother and sister intrusives, if such exist, or to areas of alteration, but to a more profound parentage giving rise to mineralizers finally emplaced in prepared environments.

The practical application of such a conception is that greater emphasis in this and other districts may successfully be placed on favorable structures and host-rock conditioning, regardless of where they occur within a given "spread" area.*

## ACKNOWLEDGMENTS

The author expresses appreciation to Walter Lyman Brown, Executive Vice-President of Knob Hill Mines Inc., for permission to present this paper. Most of the material was gathered during the last seven years while on consulting engagements in the Republic district for his company. Members of the Knob Hill staff have been especially cooperative.

Acknowledgment is made of the helpfulness of discussions with Augustus Locke covering his and Paul Billingsley's conception of ore deposits as related to the continental framework.

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4. W. Lindgren: Mineral Deposits, 2 ed., 1919.
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## DISCUSSION

AUGUSTUS LOCKE†—This paper fits, not into the customary form, but into personal views. The writer is entitled to those views for he has been in the district for eight years and has made them work. No outsider can successfully contest them.

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* An estimate of 4 miles down to the source area is given, based largely on the transverse structural pattern (Figs 6 and 7). This depth is regarded as a probable minimum. Axial openings along an 80-mile structure can safely be presumed to be effective to a depth much greater than 4 miles.

It is not held that mineral-bearing solutions extended all along the Sanpoil channel, but that such channels at appropriate places are a prime factor in ore district localization.

† Menlo Park, Calif.

The paper achieves striking unity by holding fast to the field geology of local ore occurrence and letting everything else go. Is the 1939 discovery a repeatable experiment? Will those views work a second, and a third, time? We may well remind ourselves that the repeatable experiments in ore are still limited to short range. Nothing in the Republic district from 1900 to 1938 presaged the bright future of 1939. The actual knowledge of the trap came after the discovery, and with this knowledge the engineer who knows his way can save himself the extravagence of aimless drifting and crosscutting. But he cannot yet, as they do in oil, make primary discoveries. What he will have to do in the future, as the author of this paper suggests, is to take in more territory, recognize the pattern of occurrence in the district and in the region, and prospect along big district and regional structures.

J. A. Richardson*—L. B. Wright's paper suggests several interesting comparisons with the relationships of gold mineralization to regional and local tectonics in the gold belt of Kelantan and Pahang, Malayan Union. Mr. Wright stresses the importance in ore deposition of transverse structures cutting across the regional grain of the country and ties this local picture into that of the broad regional structure of Washington and Idaho.

The regional structure of northwest Pahang and southwest Kelantan traced from west to east consists in the following six main zones:

1. The Main Range Coulisse, a granite mass some 300 miles north and south, extending from south Siam into Malacca.
2. The Main Range Foothills composed of rocks of the Arenaceous Formation, stretching from south Kelantan for 200 miles into Negri Sembilan.
3. The Sungei Serau synclinorium comprising Permocarboniferous shales and limestones together with pyroclasts of the Pahang Volcanic Series.
4. The northern extension of the Gunong Benom Coulisse, the locus of important igneous intrusions.
5. The Sungei Tanum synclinorium resembling (3) above.
6. The Gunong Tahan Coulisse composed of Triassic sediments.

Pressures were operative probably in late Cretaceous times from east or southeast westward with the result that the strata were acutely folded, commonly into isoclinals recumbent eastward. Probably at about this time the main igneous masses of the area (mainly granitic) were introduced. Gold mineralization was widespread through most of the area lying between the Main Range Foothills and the Gunong Benom Coulisse. It is believed to have been genetically related with late stage derivatives from granite magma except in the Bukit Ranjut dome where it may have derived from syenite. The location of minable deposits has been considerably influenced by local tectonics.

In the south, the Raub Australian gold mine lies southeast of the Bukit Kajang granite porphyry mass; numerous small intrusions of granite porphyry, quartz porphyry and "quartz-mica rock" (altered aplite) occur within the mining area. The gold deposits occur: (1) within two main zones of strike faulting striking NW-SE or NNW-SSE, and (2) within very narrow tensional cross fractures located between these two zones of strike dislocation.

Numerous acute minor isoclinal, drag and sigmoidal folds accompanied by much brecciation, have been developed within the strike fault zones. Ore bodies here appear to have been influenced in particular by a strong northerly or northwesterly pitch of the local structures (10 to 20° pitch is common). The richly mineralized tensional stringers average less than an inch in width. They may dip north or south or be almost vertical; important mineralization tends strongly to follow the northerly dipping fractures.

Farther north, the rocks of the Sungei Serau synclinorium appear to have been squeezed against a more stable buttress formed by the massive rocks of the Main Range Foothills. Some pincer-like crushing may have been caused by the rising of the northern end of the Gunong Benom Coulisse which was accompanied by the emplacement of the Chegar Perah-Pulai porphyry intrusion, the Bukit Ranjut syenite dome and the Sungei Jelai granite mass. The Chegar Perah-Pulai intrusion is cigar-shaped and plunges out both north and south; most of its component rocks are more or less intensely sheared and gold mineralization is widespread within them. The Bukit Ranjut intrusion is the stripped core of an acute-angled, high-sided dome (the average dips of the marginal sediments is some 70° quaquaversally); it carries considerable quantities of gold.

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* Carribean Petroleum Co., Maracaibo, Venezuela.

The Selensing and Buffalo Reed gold mines (both now closed) are located within the disturbed zone along the margin of the Main Range Foothills. The fissure patterns of both mines are essentially the same as that at Raub and local structures also tend to pitch northward.

In conclusion, the results of some five years geological surveying in Malaya before the war showed that the fold, fault and fissure patterns of sedimentary and metamorphic rocks and to a large extent the fissure systems of the igneous rocks also, could be very clearly correlated with the main features of the tectonic pattern of British Malaya. It was also observed that not only did this tectonic control exert a considerable influence upon the location of primary ore bodies, but that it influenced the deposition of placer deposits particularly in limestone bedrock due to the operative water-courses having been to a greater or lesser measure affected by jointing and fissuring or by fold humps and troughs. This form of tectonic control is particularly well exemplified in alluvial tin mines opened up on limestone bedrock in the Kinta Valley, Batang Padang and Kuala Lumpur tinfields.

# Mercury Industry in Italy

By Edwin B. Eckel,* Member AIME

(New York Meeting, February 1948)

## Abstract

THIS paper, based on brief field examination and on data supplied by the operators, records the condition of the Italian mercury industry as of March 1945, not long after Italy's collapse. Except for the once-Italian but now Yugoslavian Idria mine near Trieste, virtually all the known mercury deposits are in the Monte Amiata district in Siena and Grosseto Provinces, 75 miles north of Rome. This district contains the Abbadia San Salvatore, the Siele, the Argus, and the Cerreto Piano mines besides numerous prospects and unproductive mines. Under pressure from Germany the principal mines were highly productive during the early part of the war, but production fell off long before its end. All of the mines were damaged in some way at the German withdrawal but by September 1944 the Abbadia had resumed production and several others were only lacking power by the early spring of 1945.

The Monte Amiata district is characterized geologically by a thick series of sedimentary rocks that range from Triassic to Eocene in age and are surmounted on Monte Amiata itself by a thick cap of extrusive trachyte. Bodies of travertine and of landslide detritus, younger than the trachyte, are rather widespread. The structure is complex and little known but most of the ore deposits seem to lie along or near a major, north-trending fault or on associated cross-fractures. Cinnabar is the only important ore mineral and clay minerals are by far the most abundant products of hydrothermal alteration. A southward increase in the amount of stibnite, relative to cinnabar, suggests horizontal zoning. The grade of ore ranges from 0.60 to 3.0 pct and the reserves, as compared with those in American mines, are enormous. Even in the face of war-time scarcities, the mines are remarkably self-sufficient and labor is plentiful, but the costs of production are estimated to be not greatly below those of the better American mines. Notes on the principal mines, on the mining and metallurgical methods, and on the manufacture of synthetic cinnabar are included in the paper.

## Introduction

With but few brief interludes, Spain and Italy, including the Idria mine near Trieste, which has passed from Austrian to Italian to Yugoslavian hands in the past 3 decades, have led the world in the production of mercury, or quicksilver, for many centuries. Despite their enviable position with respect to the supplies of a metal so essential in both war and peace, there is a notable lack of up-to-date information in America on the mercury industry of either country. Duschak and Schuette[1] and Schuette[2,3] give by far the best summaries in English known to the writer, but the latest firsthand report cited by them on the Italian mines was published in 1915.[4] Having been deeply interested in the geology and economics of mercury for several years, the writer naturally felt most fortunate in being able to obtain enough firsthand information on the Italian industry to fill at least a few of the gaps in our knowledge. The opportunity came during the war, soon after this industry, together with the rest of Italy, had capitulated to the Allies. Many of the facts presented here are thus definitely "dated," but others are of less transitory nature, and even the dated facts should be of interest in showing the strength and

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Published by permission of the Director, U. S. Geological Survey. Manuscript received at the office of the Institute June 18, 1947. Issued as TP 2292 in MINING TECHNOLOGY, January 1948.

* Chief, Section of Engineering Geology, U. S. Geological Survey, Denver, Colorado.

[1] References are at the end of the paper.

resilience of Italian mines in the face of adversity and destruction.

While serving as a civilian Technical Representative in the European Theater the writer was temporarily detached from duty with the Intelligence Division, Office of Chief Engineer, and assigned to investigate the mercury industry of Italy. On reporting to Allied Force Headquarters, Mediterranean Theater, he was attached to the Mining Division, Industry Subcommission of the Allied Commission for Italy. This assignment was made at the instance of the Metals and Minerals Ad-

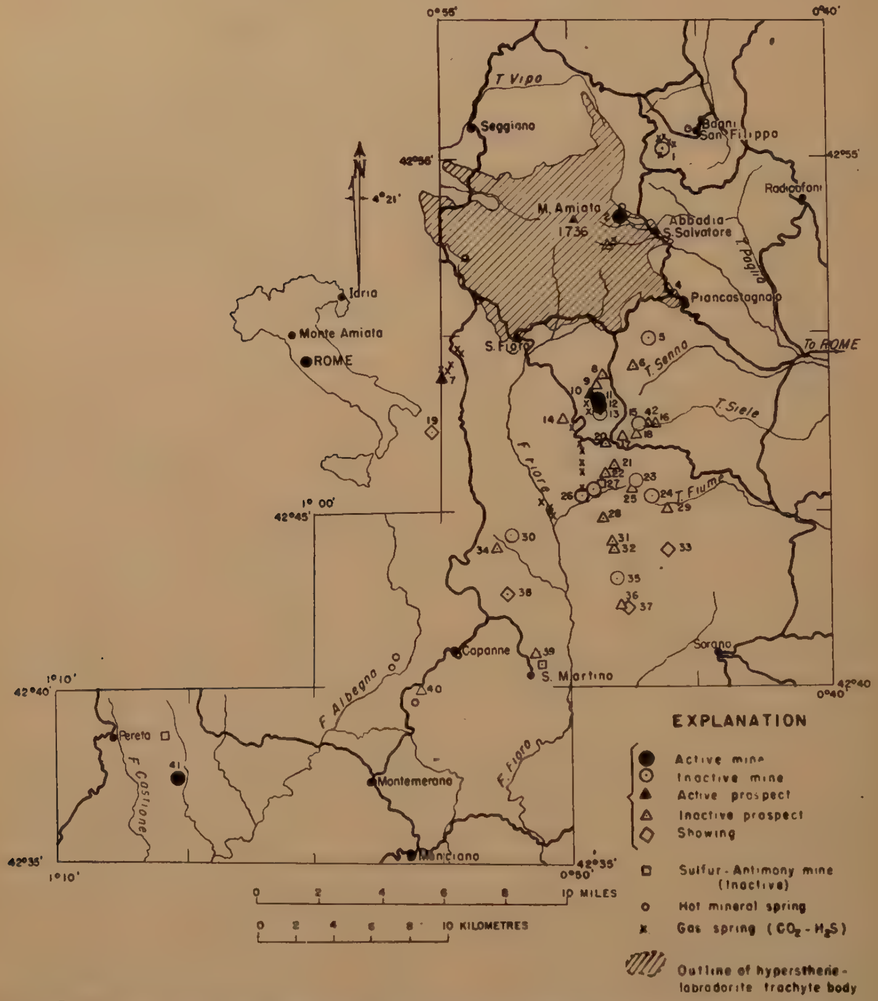

FIG 1—INDEX MAP, SHOWING LOCATION OF PRINCIPAL MINES AND PROSPECTS IN THE MONTE AMIATA MERCURY MINING DISTRICT, ITALY.

Compiled from various sources by Edwin B. Eckel, March 1945.

Base sketched from G.S.G.S. No. 4164 Italy 1:100,000, sheets 129 (Santa Fiora) 135 (Orbetello) and 136 (Tuscania).

(See opposite page for explanation of numbers) visory Board, Army Service Forces and of the U. S. Geological Survey, both of which posed certain specific questions on the status of the industry. Some of the data needed to answer these questions were obtainable only through the wholehearted cooperation of the private Italian companies. They were given for confidential use of the United States Government and therefore are not included in this paper.

Most of the information contained here relates to the mines of the Monte Amiata district in the Tuscany Region, Siena and Grosseto Provinces. The only other quasi-Italian mine of consequence is the Idria mine in Gorizia, near Trieste. It was still in enemy hands in March 1945, when this investigation was made, and belonged to Yugoslavia as of mid-1947.

About one month was devoted to the study, including preparation of official reports and collection of data from company and Italian Government officials in Rome. Of this time, only six days were spent in actual examination of the mines. Three of the four principal mines of the Monte Amiata district, the Abbadia, Siele, and Argus, were visited, as were a number of smaller properties.

## GEOGRAPHY

Except for the Idria mine, in Gorizia Province, northeast of Trieste, virtually all the known mercury deposits of Italy are in the Monte Amiata district in Siena and Grosseto Provinces (Fig 1).

This district, about 75 miles north of Rome, is accessible over State and provincial roads that by March 1945 had been placed in excellent condition after the enemy's retreat except for many bridges still under repair. The Monte Amiata depot of the Grosseto-Siena railway line, then being repaired, is about 30 miles north of the Abbadia mine.

The altitudes range from about 150 meters (approximately 492 ft) at the Cerreto Piano mine to about 900 (approximately 2953 ft) at the Abbadia. Monte Amiata, on whose slopes most of the mines are situated, is 1734 meters (approximately 5689 ft) high. The climate is relatively mild and permits year-round work.

The mineralized area of the Monte Amiata district is large. As shown on Fig 1 the principal mines are concentrated within a north-south belt about 35 kilometers long and 8 kilometers wide (approximately 22 by 5 miles) that extends along the east slope of

1. *Bagni S. Filippo or Pietrineri* (*A*)[1]
2. *Abbadia S. Salvatore* (*A*)
3. Cipriana
4. Infovina
5. Casa Paolo
6. della Senna
7. Monte Labbro (*A*)
8. Cerrone
9. Renaioli (*B*)
10. Grande Putizza, or Nibbio (*C*)
11. *Argus or dell Abetina* (*B*)
12. *Solforate del Siele or S. Rosselli* (*C*)
13. *Solforate Monte Amiata or S. Schwartzenberg* (*A*)
14. Selva
15. *Siele* (*C*)
16. Cesari, Castoloni or Barzellotti (*A* and *C*)
17. Paicciole or Valloni
18. Rigacci
19. Solforate
20. Poggio la volpe
21. Borghetto
22. Poggio Paulorio
23. *Cornacchino* (*A*)
24. *Sezzione Poggiali or Riparelle* (*A*)
25. Poggio Felioso
26. *Morone* (*A*)
27. *Selvena or Dainelli* (*A*)
28. Casa Cappelletti
29. Poggio Folaca or Putoni
30. *Cortevecchia* (*A*)
31. *Casa Testi* (*Schwarzenberg*)
32. Casa Testo (Menicanti e Cia)
33. Fonte del Ballino
34. Salvegnana
35. *Reto or Montebuono*
36. del Segno
37. Casa Gatti
38. Casa Zammarchi
39. S. Martino di Monticchio
40. Saturnia
41. *Cerreto Piano* (*C*)
42. Carpine (*C*)

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[1] *Ownership or Control*
(*A*) Soc. An. Miniera Monte Amiata
(*B*) Soc. An. Miniera Argus
(*C*) Soc. An. Stabilimento Minerario del Siele an-Campine

Monte Amiata and south therefrom. One active mine, the Cerreto Piano, is fully 30 kilometers (approximately 19 miles) southwest of this belt, however, and several inactive mines and prospects are well outside the main zone. The entire district is thus more than 500 square miles in area.

## HISTORY AND PRODUCTION

*Idria District.*—The Idria deposit has been worked almost continuously since early in the 15th Century and is second in the world to the Almaden district, Spain, in total production. Its output in recent years has declined both in quantity and grade, but it still contains larger and richer known reserves than any American mine. Its deepest shaft, the Jozefor, was reportedly 387 meters (approximately 1270 ft) deep in 1944 and had 15 levels. Partisans sabotaged the mine by destroying the pump on the 14th level. This flooded the mine up to the 11th level, at 257 meters (approximately 843 ft) depth, below which lay the richest ore reserves.

TABLE 1—*Mercury Production in Italy,* 1940 *to* 1944[a]
FLASKS OF 76 LB.

| District | 1940 | 1941 | 1942 | 1943 | 1944 | Totals |
|---|---|---|---|---|---|---|
| Monte Amiata district | 82,278 | 79,320 | 64,714 | 48,661 | 21,327 | 296,300 |
| Idria district | 13,800 | 14,600 | 11,050 | 8,400 | 1,670 | 49,520 |
| Totals | 96,078 | 93,920 | 75,764 | 57,061 | 22,997 | 345,820 |

[a] Compiled from company records but subject to adjustment when official government figures become available.

*Monte Amiata District*—The surface outcrop of the Siele mine was worked by the Etruscans several centuries B. C., but the Monte Amiata district's modern history did not begin until 1868, when the Siele was "rediscovered" and opened. The original Siele (No. 15 on Fig 1) is now idle and is possibly exhausted, but a nearby mine on the same mineralized zone, the Solforate del Siele (No. 12), has taken its place. For convenience, the latter is called Siele throughout this report.

The Abbadia San Salvatore mine, now the most productive mercury mine in Italy, was discovered in 1894 and began large scale production about 1901. During the past 40 years the Abbadia, Siele, and Idria together have been responsible for more than 40 pct of the world's mercury supply.

The district contains several smaller mines, some of them with sizeable reserves, as well as several promising prospects; but as long as the ore reserves of the principal mines remain so large and rich there is little incentive for opening them or for further search for new ones.

Under pressure from Germany to supply her war needs, the Italian mines were highly productive during the early part of the war, but as shown in Table 1, production began to fall off somewhat in 1942. How much of this was caused by disaffection on the workers' part and how much to the owners' dissatisfaction with the German requisitioning methods is not clear. Production ceased at the time of the German withdrawal and in March 1945, had been resumed only by the Abbadia mine.

## GEOLOGY OF MONTE AMIATA DISTRICT

### *General Statement*

The writer had opportunity to make only the most casual examination of the geology either in the mines or in the district as a whole. Most of the following remarks are therefore taken from other sources than direct observations.

The geology of the district has been mapped on a scale of 1/25,000 by De

Castro,[4] whose report contains detailed descriptions, cross sections, and maps of all the mines that were known at the time of his study. This is an excellent report and is relied upon by all the mining men of the district. Its chief shortcomings from the present point of view are that it is now out of date, that it contains but few data on grade, reserves or production, and that it stresses lithology almost to the exclusion of structural geology, both in the general description and in discussions of the ore deposits. This latter distinction characterizes the thinking of the present-day operators; they recognize and map the local rock units with the greatest detail, but none of them seems to have more than generalized ideas on structure. Schuette[2] gives references to earlier reports and a summary of the geology.

### *Igneous Rocks*

A large body of coarsely crystalline hypersthene-labradorite trachyte, which makes up the greater part of Monte Amiata itself, is the only igneous rock of importance in the district. There are a few small plugs of post-Pliocene serpentine throughout the area, as well as some basaltic flows and dikes in the southeastern part of the area shown on the index map. Neither kind of rock has any apparent connection with the mercury deposits whereas the trachyte may well have a close genetic relationship with the ore. The age of the trachyte is not known, but it is certainly late Tertiary or even younger. On the maps of the Italian Geological Survey it is shown as an extrusive mass, with a nearly flat base, overlying the eroded surface of the Eocene and older sedimentary rocks. The Italian maps also indicate a feeder channel beneath the center of the trachyte mass, marked by a very considerable displacement of the sedimentary rocks. The existence or location of this feeder is probably entirely hypothetical. Whether the trachyte was originally a laccolith or was extruded on the surface is possibly open to conjecture, but what evidence the writer saw on the eastern and southern side of the body points to an extrusive origin.

There is evidently some variation in the rock from place to place. Clarence S. Ross of the Geological Survey examined two specimens of the "trachyte." He describes one specimen, taken from the surface above the Abbadia ore deposit, as a "rhyolitic rock, composed of glass with perlitic structure enclosing phenocrysts of quartz, sodic plagioclase, augite and biotite. A little magnetite and pyrite is present." The other specimen, taken from one of the Abbadia company's rock quarries on the slope of Monte Amiata, was found by Ross to be a "trachytic rock composed of a very fine grained groundmass and phenocrysts of quartz, plagioclase, augite and biotite. The groundmass is in part isotropic but contains small phenocrysts of plagioclase and biotite."

### *Sedimentary Rocks*

Sedimentary rocks, consisting of shale, sandstone, marl and limestone, underlie the greater part of the district. They range from Triassic to Quaternary in age, but those of Eocene age are the most widespread. Cinnabar deposits are known in all but the Quaternary rocks, but by far the most important ones are in the Eocene.

The stratigraphic units are sufficiently distinctive in lithology to permit ready identification by local miners and operators who have had long experience with them. The writer's too brief acquaintance with them left him with the impression that it would be easy to misidentify some of the units, particularly where they are strongly altered, unless the structural relations were mapped in detail. The generalized stratigraphic section of the Monte Amiata district is as follows:

*Quaternary*—

Recent and older: Travertine, hot spring deposits, at Bagni San Filippo and in southwestern part of district. Older travertine quarried extensively for ornamental stone; younger material, which contains sulphur, is used for viticulture.

Older: Landslide detritus in large masses along slopes of Monte Amiata. Some landslides almost certainly pre-Pliocene in age.

*Tertiary*—

Pliocene and Miocene:

Small lake beds, mostly clay but containing a few peat deposits in depressions on surface of older rocks. Clays are mined for brick and tile manufacture. Contain one mercury prospect in southwest part of district.

Eocene:

1. Upper sandstone (arenaria superiore) generally contains a few thin lenses of limestone.
2. Marly limestone and shale (calcari marnosi e schisti argillosi), with few lenses of nummulitic limestone and of sandstone. "*Galestri*" of the local miners. Contains most of the mercury ore in the Abbadia mine and part of that in the Siele mine.
3. Lower sandstone and sandy shale (arenaria inferiore). Contains part of the mercury ore in the Siele and Argus mines.
4. Nummulitic limestone (calcare nummulitico), variegated and brecciated light gray granular limestone with nummulites. Intercalated with lenses of compact marl.

*Cretaceous*—

Senomian in age: Clay shale, vari-colored, with traces of copper minerals. Red-green manganiferous jasper. Some red-green and gray limestone with vari-colored veining.

*Jurassic*—

Lias:

Upper marly limestone and shale. Middle gray limestone with some red limestone containing ammonites. Lower massive limestone with gastropods.

*Triassic*—

Rhetic (Retico) in age: Gray, massive limestone with Avicula contorta and other fossils; marly shale with Bactrilli; dolomitic limestone with Megalodus.

### *Rock Alteration*

The trachyte, most of which is relatively unaltered, with clear glassy feldspars, is locally discolored and partly altered to clay minerals. This alteration seems to be the result of hot spring activity rather than of weathering, but its origin, as well as the relationship to the ore deposits, deserves further study. Pyrite is widespread in the trachyte and is much more abundant there than in the sedimentary rocks.

Clay minerals are by far the most abundant alteration products in the sedimentary rocks. The only others noted are a little pyrite and marcasite, with some calcite. Hasty examination indicates that the clay alteration is confined to hot spring areas and to faulted or crushed zones that contain ore deposits. The clay, which is usually dark gray to black and very sticky, is doubtless formed partly by attrition of the argillaceous country rocks. Unquestionably hydrothermal alteration has also played a large part in its formation for even massive limestone is very clearly converted to clay in many places. Clarence S. Ross examined a number of specimens of ore and of altered rocks and reports that the clay minerals are micaceous and appear to be of the hydrous mica type.

### *Springs and Gaseous Emanations*

As shown on Fig 1, hot springs and gaseous emanations are widespread throughout the district. There are also many cold springs near the edges of the trachyte mass which yield iron-sulphate bearing waters.

Carbon dioxide and hydrogen sulphide are the principal dissolved constituents of the hot springs, and some of them are still depositing travertine and sulphur. The gas springs (putizza) yield warm gas, some of it under relatively high pressure, which consists of 95 to 98 pct carbon dioxide and 2 to 5 pct hydrogen sulphide. They are obviously allied to the hot springs, both in space and in origin. The rocks near most of the hot springs are discolored and strongly altered to clay minerals.

The mines are not greatly troubled with hot water, but gas, similar in content to that of the putizza, is a real problem in most of them. Although hot springs or gas emanations are found near some of the mines, hot spring areas generally are not regarded as favorable prospecting areas by the mine operators.

### *Structural Geology*

The geologic structure of the Monte Amiata district is an almost complete enigma to the writer. It is unquestionably complex, at least in places, for scattered observations of fairly good exposures throughout the district failed to give any picture of structural continuity within the sedimentary rocks. There is certainly some folding, for the rocks in different exposures are at widely divergent attitudes. There is also much faulting, as illustrated, for example, by the enormous downdropped block of sandstone in the Siele mines (Fig 2), and by the juxtaposition of Triassic and Eocene rocks in the Selvena mine. There is also good evidence that the entire Abbadia mine is in a fault zone, for the rocks encountered there range from Cretaceous to Eocene, the blocks dip at all conceivable angles, and the ore is clearly derived from breccia, most of whose fragments are now altered to clay (Fig 3 and 4).

The alignment of mercury deposits, hot springs, and gas emanations shown on Fig 1 strongly suggest the existence of a wide, major fault zone that trends nearly due north and extends from near the southern part of the district at least as far as Bagni San Filippo. Some evidence, presented under the discussion of ore deposits, indicates that this fault zone is in part older than the trachyte. The structural relation of the original Siele, the Cerreto Piano, and other deposits to the suggested major fault zone is not known.

## ORE DEPOSITS

*Mineralogy and character of ore*—Cinnabar is the principal ore mineral in all the Monte Amiata deposits. Native mercury is occasionally seen but always in very small quantities. The writer suspects the presence of considerable quantities of meta-cinnabar, both from the generally dark color of most of the "cinnabar" and from the appearance of pannings. Most of the cinnabar is dark red and distinctly crystalline. Well-formed crystals are scarce, except in fracture-linings of limestone blocks and in a few specimens of limestone-replacement ore. In the Siele mine, some of the cinnabar is light red and powdery, and some is massive, made up of radiating crystals that are strongly suggestive of travertine replacement. Both kinds are relatively scarce. Small concretionary nodules, or "strawberries" of cinnabar occur in some veins; these often contain a central nucleus of native mercury. Pyrite and marcasite are the only other metallic minerals in the ore from most mines, but stibnite is associated with cinnabar in the Selvena and elsewhere. Pyrite and marcasite have about the same distribution as cinnabar, and also occur in the trachyte. The only other introduced minerals in the

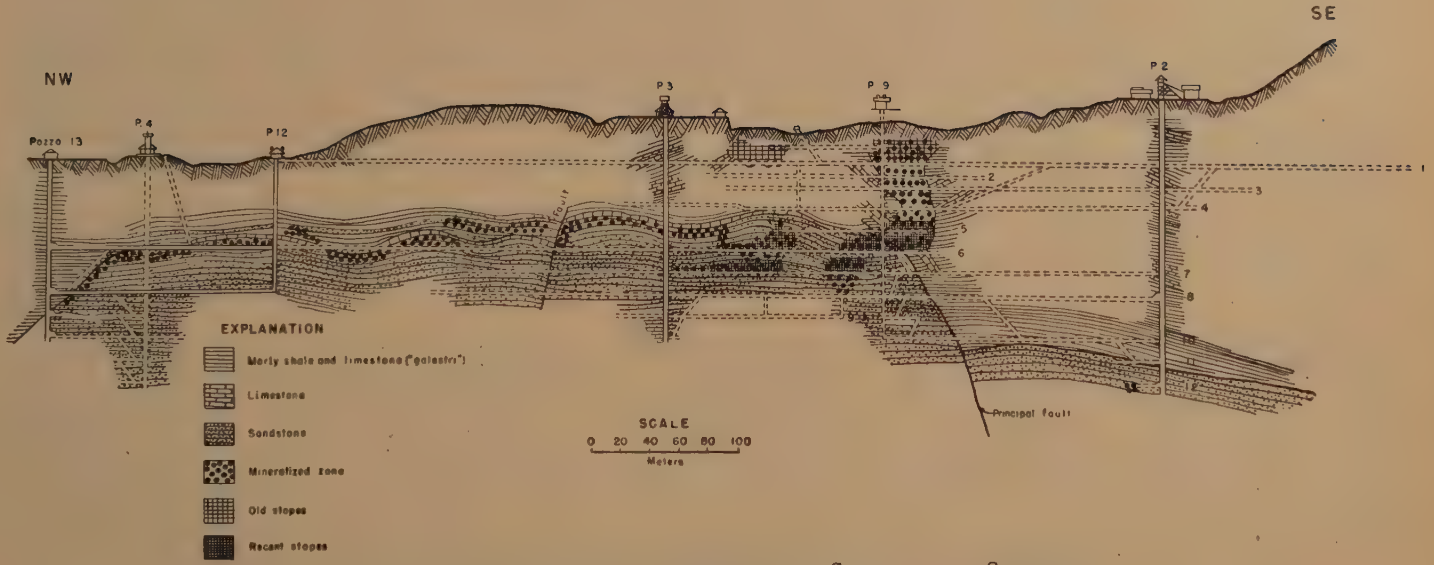

Fig 2—Generalized geologic cross-section of Solforate del Siele mine. (From company map)

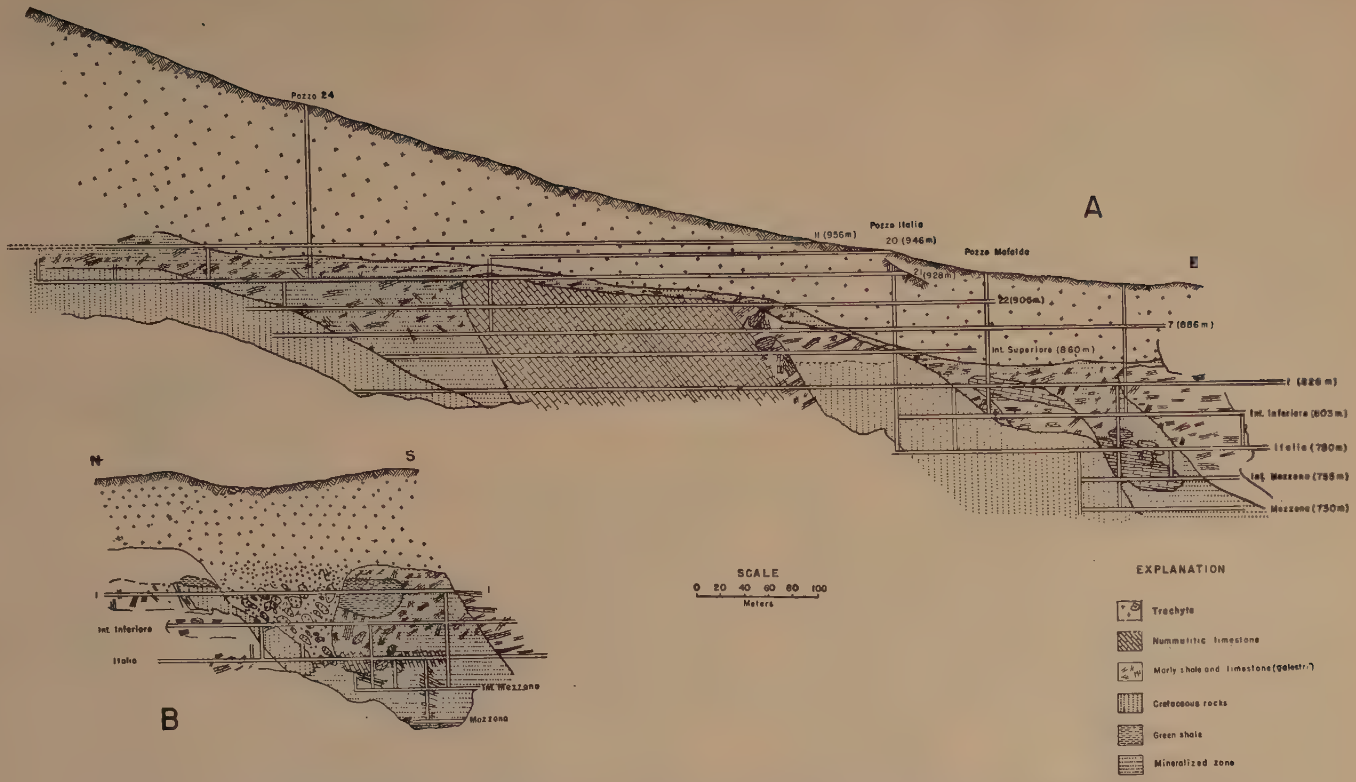

FIG 3—GENERALIZED GEOLOGIC CROSS-SECTIONS OF ABBADIA SAN SALVATORE MINE, SHOWING CHARACTER OF ORE BODIES.
*A*—West-East section through central part of mine.
*B*—North-South section near eastern end of mine.
(From company map)

ore, other than clay, are calcite, which forms veinlets and lines rare vugs, and gypsum. The latter may be of secondary origin, but crystals of gypsum enclosing cinnabar are seen in places. Even these may represent grains of cinnabar that were enclosed in the growing gypsum crystals rather than contemporaneous deposition of primary gypsum and cinnabar.

The typical ore of the Abbadia mine is dark gray to black clay ("polpone") that fills spaces between fragments and residual boulder-like masses of unaltered nummulitic limestone (Eocene). The limestone is

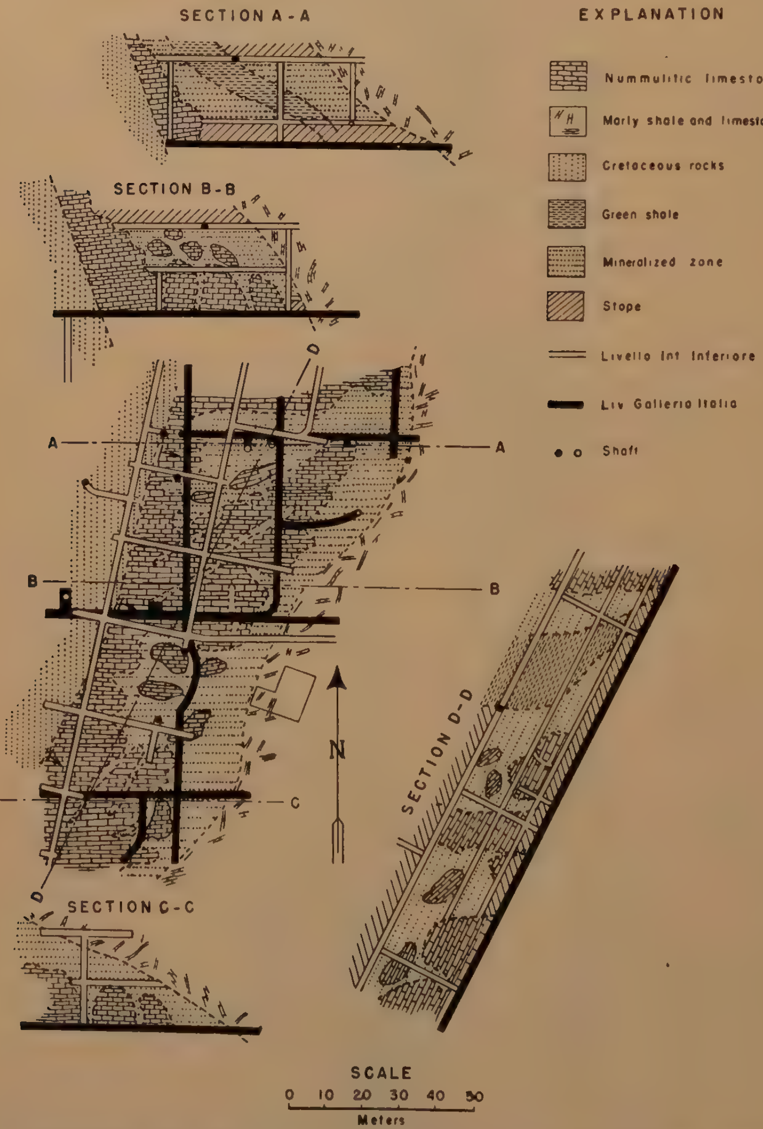

FIG 4—DETAIL PLAN AND SECTIONS OF PART OF ABBADIA SAN SALVATORE MINE, SHOWING CHARACTER OF THE ORE BODIES.
Typical of records maintained for all workings. (From company map)

seldom ore-bearing even where completely altered to sugary or clayey material. Some limestone blocks contain facings of cinnabar on fractures or on surfaces of the blocks, but never in sufficient quantity to constitute ore. The dark clay ore contains streaks and blebs that are visibly rich in fine-grained cinnabar, as well as nodules one-half inch to several feet in diameter that consist of 40 pct or more of cinnabar cementing unreplaced fragments of shale, sandstone and limestone. Vugs in this massive ore contain some cinnabar crystals, but more commonly they are lined with films of gray clay. Even clay that contains no visible cinnabar will often "pan" several percent of the mineral. Much of the richest ore is on or near the borders of limestone blocks.

Some of the Siele and Argus ore is like the clayey ore of the Abbadia, but much of it consists of veinlets of cinnabar and of partial replacements of sandstone and limestone. The limestone ore is never as rich as that in the sandstone. In the latter, the cinnabar selectively replaces calcareous cement between the sand grains and avoids sandstone with siliceous cement. Some of the calcareous sandstone ore is very rich.

*Structural relations of the ore*—Without knowing anything of the geologic structure of the Monte Amiata district, it is not possible to draw any valid conclusions as to the structural control of the ore bodies. Nevertheless, it may be permissible to present a few thoughts on the subject so long as they are clearly labeled as guesses.

Schuette[2] has presented an excellent critical summary of the literature on the Italian deposits and describes and illustrates numerous examples of structural control of individual ore bodies. His paper should be read collaterally with this one for a more complete picture, but the writer's knowledge seems an inadequate basis for synthesis of his own and Schuette's conclusions.

There is a fair amount of evidence that the major deposits of the district are controlled by a strong, north-trending fault zone that extends the entire length of the district. In places, as in the new Siele mine, the rocks are disturbed in only a comparatively narrow zone along this fault. The deposits of both the old and new Siele mines, as well as of the Argus, may well have been controlled by the intersection of the north-trending major fault with a well-defined series of lesser cross-faults that trend northwest.

Elsewhere, particularly at and near the Abbadia mine, the major fault zone is wide and the sedimentary rocks within it are jumbled and have been strongly altered by hydrothermal solutions. The impermeable nature of the clayey ore, together with the presence of clay minerals in vugs of cinnabar, strongly suggest, if they do not prove, that the clay and cinnabar are contemporaneous, and that both were formed by the same hydrothermal solutions. Had the clay been formed first by alteration of a breccia, it is difficult to see how or why the cinnabar solutions could have circulated through it freely enough to deposit the rich ores. On the other hand, if the cinnabar had been deposited first, one might suppose that the cinnabar would be attacked or moved by solutions capable of complete alteration of the matrix. The trachyte itself is locally altered, but it is not noticeably fractured, even directly above the Abbadia deposit. This fact would suggest that the trachyte is younger than the major faulting were it not for the fact that the base of the trachyte in the Abbadia mine is everywhere marked by 0.5 to 5 meters (approximately 1 ft 8 in. to 16 ft) of black clay gouge. Moreover, fragments of altered trachyte have been found in the lower part of the ore body, far below the trachyte contact.

For the present, it seems reasonable to suppose that the Abbadia deposit was controlled by a wide, north-trending crush zone that was formed in the sedimentary rocks before the trachyte was emplaced. The writer would explain the clay at the trachyte-sedimentary contact partly as a result of movement during and after the trachyte extrusion and partly as a result of post-trachyte hydrothermal alteration. The fragments of trachyte in breccia below the contact (Fig 3*B*) may possibly mean that the fault zone was one of the feeder channels for the trachyte. Whether or not this is true, it seems entirely possible, if not probable, that the chaotic nature of the rocks, the presence of mineralized blocks of trachyte at depth, and the abundance of clay minerals are all the result of large scale subsidence that accompanied and resulted from the mineralization processes. The part played by the trachyte cap in guiding the hydrothermal solutions is not known, but it may well have had a very considerable effect in diverting the solutions and thus localizing the deposits. The present day hot springs and gases can be regarded as the waning stage of the process of alteration and ore deposition.

*Age of cinnabar mineralization*—According to Signor Masobello, one prospect in the extreme southwest end of the district is in Pliocene sediments. This, and the close apparent relationship of the cinnabar deposits with hot springs emanations, suggests that all the deposits are of late Pliocene or post-Pliocene age. Their relative youthfulness is further attested by the fact that delicate crystals and powdery cinnabar have not been crushed or smeared by post-mineralization movements.

*Zoning*—There is some evidence of horizontal zoning in the Monte Amiata district. That is, the proportion of antimony in the ores increases from north to south. The Bagni San Filippo, Abbadia and other deposits in the northern end of the district contain very pure cinnabar with no metallic impurities. Some of the deposits farther south, such as the Selvena, contain stibnite mixed with the cinnabar, though seldom in quantities sufficient to interfere with the extraction of mercury. Near the Selvena and still farther south are several antimony mines where stibnite is the chief mineral. Pyrite and marcasite seem on hasty examination to be more abundant in the Siele, Argus and Selvena ores than in the Abbadia, but this relationship may be more apparent than real.

## RESERVES

There was no possibility for the writer to make independent reserve estimates within reasonable time and expense. The Argus workings and most of the smaller mines were inaccessible because of gas or caving and the ore bodies of the larger mines are never exposed for distances of more than a few meters at a time because of heavy ground. The company reserve estimates cannot be disclosed, but it can be stated that the reserves of measured ore in the Monte Amiata district, blocked out and ready for mining, are enormously larger than are similar reserves in United States mines. The Abbadia and Siele mines, whose reserves are comparable in size, together contain by far the greater part of this blocked out reserve. The Argus mine in the Monte Amiata district, as well as the Idria mine in Gorizia have far smaller reserves than the two last-mentioned, but even these have larger known reserves than any American mine. The average grade of ore in the various mines ranges from 0.6 pct at Idria through 1.0 pct at Abbadia to 3.0 pct at the Siele or 12 to 60 lb of mercury to the ton.

None of the Italian operators attempt to estimate probable and possible ore, but it is significant that the larger ones make a standard practice of developing at least as much new ore as they extract each year. All evidence points to their ability to continue this practice for some decades.

The company estimates are made up from detailed measurements of small development blocks. Most of these range in size from 4 by 5 by 10 meters (approximately 13 by 16 by 33 ft) to 12 by 15 by 70 meters (approximately 40 by 49 by 230 ft). Grade is determined by daily panning and assay of samples from each working face.

About 15 to 20 samples are panned each day by expert panners, checked by the mine superintendent. Daily samples are assayed for control and allowances are always made for the weight, in each block, of barren rock that would be discarded on mining

### MINES

*Abbadia San Salvatore Mine*—The Abbadia San Salvatore mine, near the town of the same name, is the farthest north of the productive mines in the Monte Amiata district (Fig 1). It is owned by The Societa Anonima Mineraria Monte Amiata of Rome, which holds a concession in perpetuity on the Abbadia mine and also owns concessions on at least eight other abandoned or inactive mines in the Monte Amiata district (Fig 1). In 1940 it took over the formerly State-owned Idria mine but as this mine is in the controversial Trieste area, the company's future tenure would appear doubtful. In 1945 the Monte Amiata company was 53 pct government-owned or controlled.

The Abbadia was discovered in 1894, but produced comparatively little mercury until about 1901; since then it has been the largest and most consistent producing mine in Italy. Its outcrop was discovered by a peasant and the property was soon taken up by a German engineer, with German capital, who opened a large surface cut. This was later followed by discovery of rich ore below the surface and underground operations were commenced. The discovery must have been one of nearly pure chance, for almost the entire surface area of the Abbadia deposit is hidden by unmineralized trachyte. This fact, as well as the presence of mineralized springs in the trachyte, lends hope for the ultimate discovery of other mercury deposits beneath the trachyte.

The Abbadia was the only mine at Monte Amiata that was seriously damaged by the Germans when they withdrew. The surface plant was completely destroyed, but the main hoist and much portable equipment were saved for they had been dismantled and hidden underground. After the "liberation" the company rebuilt some of its furnaces and began operating in September 1944, producing mercury partly from bricks of the wrecked furnaces and partly from ore that could be mined by hand methods. The company's private hydroelectric plant was undamaged but all power beyond the minimum needed for maintenance was allocated to other essential users.

The extent of the mine workings and the shape, size, and general character of the ore bodies are indicated on Fig 3. The explored ground occupies a roughly rectangular area that trends a few degrees north of west and is 2000-ft wide by 4000-ft long. Fig 4 shows in somewhat more detail the characteristics of a typical ore body. It is unfortunate that structural features such as faults and attitude of bedding have not been recorded, but it is obvious that the rocks beneath the trachyte cap can only be interpreted as a gigantic breccia containing blocks of varying size. Locally (Fig 3*B*) the trachyte itself is involved in the breccia. The ore deposit is blanket-like in general form and here consists largely of cinnabar-bearing "galestri," or crushed shale that has been much altered to sticky clay, and that contains blocks of more or less altered limestone. The ore body is limited upward by the base of the trachyte and good ore is known to extend at least 130 meters (approximately 426 ft) vertically below the trachyte. The body has an irregularly pipe-like form and has not yet been bottomed, though the ore shoots are smaller in lateral extent on the lower levels than they are closer to the surface.

*Siele Mine*—The surface outcrops of the old Siele mine, about 10 kilometers (approximately 6 miles) south of the Abbadia, (Fig 1, No. 15), were worked by the Etruscans several centuries before Christ. Staghorn picks, stone hammers, jars of powdered cinnabar (presumably used for paint), and leather ore bags have all been found in the workings. The Etruscans are said to have been stopped by the abundant gas; the mine was not worked again until 1868. From then until about 1933 it was highly productive, but was finally abandoned by the company in favor of the nearby Solforate del Siele, (Fig 1, No. 12), which contains richer ore and less gas. The company is now exploring the Grande Putizza prospect near the Siele (Fig 1, No. 10) as well as a fault block of mineralized sandstone, 1½ kilometers (approximately .9 mile) wide, 5 kilometers (approximately 3 miles) long and of unknown depth, that extends between the old and new mines.

The Siele is owned by the Societa Anonima Stabilimento Minerario del Siele of Rome, a privately owned corporation, which holds two contiguous concessions in perpetuity on Carpine Solforate (Grosseto Province) and Alveo Siele (Siena Province). It pays the Government an annual concession rent of 10,000 lire, plus a royalty of 2,000 lire per flask. The mine and plant were left undamaged by the Germans but the private hydroelectric plant was destroyed. The mine was thus unable to resume operation, but it has a Diesel electric plant sufficient for pumping and partial maintenance, though not for ventilation.

As shown on Fig 2, the upper levels of the new Siele mine, or Solforate, are in marly limestone and shale (galestri). The rocks and ore are similar to those of the Abbadia except that the rocks are reasonably well-bedded and not as strongly brecciated or faulted as in the Abbadia. The galestri is underlain with nearly horizontal contact, by massive fine-grained sandstone, some of which has limy cement and some siliceous cement.

A strong fault (Faglia Principale) cuts through the mine. It strikes due north, dips 60° to 70° east, and drops the sandstone many meters down on the east side, thus forming the boundary of the mineralized block between the old and new mines. The upthrown block, and possibly the downthrown one, is cut by a series of parallel, nearly vertical faults that trend northwest. These are followed by the main drifts in the mine. Most of those seen were very tight, with little breccia and few grooves or other evidences of movement.

Ore occurs along and near the Faglia Principale as well as in favorable beds along the northwest faults. The ore in the upper part of the body usually follows limestone contacts or surrounds blocks of limestone in the marly rocks. Most of the deeper ore is in sandstone, where it selectively replaces the calcareous cement and coats fractures. Individual ore bodies are comparatively small and streaky, but their grade is high and they are numerous.

*Argus Mine*—The Argus or Abetina (Fig 1, No. 11) which is actually a continuation of the Solforate del Siele, was not discovered until 1925. Since that time it has produced a total of 50,000 flasks, with a maximum annual production of 7000 flasks. It is privately owned by Societa Anonima Mineraria Argus of Milan, which holds the land on concession from the Government. The mine, which buys its power, was left intact by the Germans, but was severely looted by certain Allied troops. They took most of the remaining mercury stocks, destroyed maps and other records and damaged both plant and shaft. The main shaft had been repaired by March 1945, using Diesel generated power, but the workings were inaccessible because of gas and presumably badly in need of maintenance.

Fig 5, sketched from a hasty glance at the remaining company mine maps, shows the essential features of the geology. A northwest-trending fault that bounds the deposit on the northeast side drops the massive sandstone country rock at least 60 meters (approximately 197 ft) to the northeast; the sandstone has not been explored on that side.

Individual ore bodies follow vertical, northwesterly fractures parallel to the boundary fault. They are usually 2 to 5 meters (approximately 7 to 16 ft) wide, but locally enlarge to as much as 50 meters (approximately 164 ft) wide and 50 meters high. Rich ore is often found at or near the contact between sandstone and black limestone, but it is never found in the limestone itself.

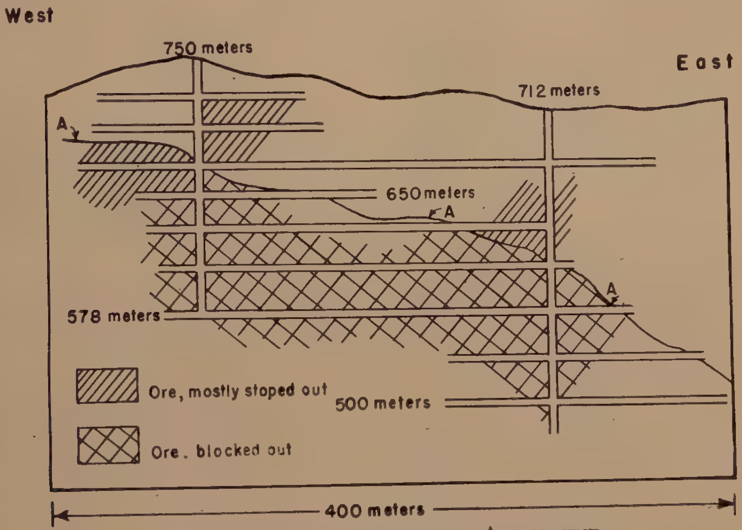

Fig 5—Sketch cross-section of Argus mine.

Line *A* represents approximate contact of marly limestone and shale ("galestri") above, with massive sandstone below. The "galestri" contains some siliceous sandstone lenses; sandstone contains a bed of black, fissile limestone at or near top.

*Cerreto Piano Mine*—The Cerreto Piano mine (Fig 1, No. 41), is owned by Societa Italiana Anonima Mercurio ("SIAM") which is in turn owned by the Siele Co. It is of interest chiefly because it demonstrates the existence of commercially valuable deposits many kilometers to the southwest of the district's principal mines. According to Ing. Carolos Spirek of the Siele Company it normally employs 110 workers who produce 150 to 200 flasks per month from 1.5 pct ore. Known reserves amount to only 5000 to 6000 flasks in 1.5 pct ore but further development of reserves is underway. The mine was undamaged by the Germans but lacked power for maintenance or production.

### Mining and Timbering Methods

The Abbadia mine, where nearly all the ground is very heavy, uses a horizontal cut and fill method of stoping described below. The Siele and Argus mines use the same system in part, but in stronger ground they are able to work their vein-like ore bodies by overhand stoping, using comparatively little timbering or other support.

In general, an Abbadia stope is worked by taking out thin horizontal slices, backfilling with waste from the same stope, from other parts of the mine or from the surface. After a slice 2 to 3 meters (approximately 7 to 10 ft) in height has been mined out and filled, a new sublevel is opened above the filled slice and the operation is repeated, the ore being dropped through raises to the main level. Fig 6*A* shows various stages in the stoping of an ore body. The length of a "stope-drift" used to extract ore from the horizontal slices is never more than 7 meters (approximately 23 ft) long nor more than 2.2 meters (approximately 7 ft) wide. Where the ground is too heavy to support a double stope width (4 to 4.5 meters or approximately 13 to 15 ft) the method shown in Fig 6*B* is adopted. During actual stoping the ground is supported by orthodox cap and post timbering. About 40 to 50 pct of the timber is recovered on withdrawal from a stope. A method of vertical slice stoping was under consideration by the management in March 1945. This method, sketched in Fig 6*C* would, it was hoped, permit more economical work in heavy ground.

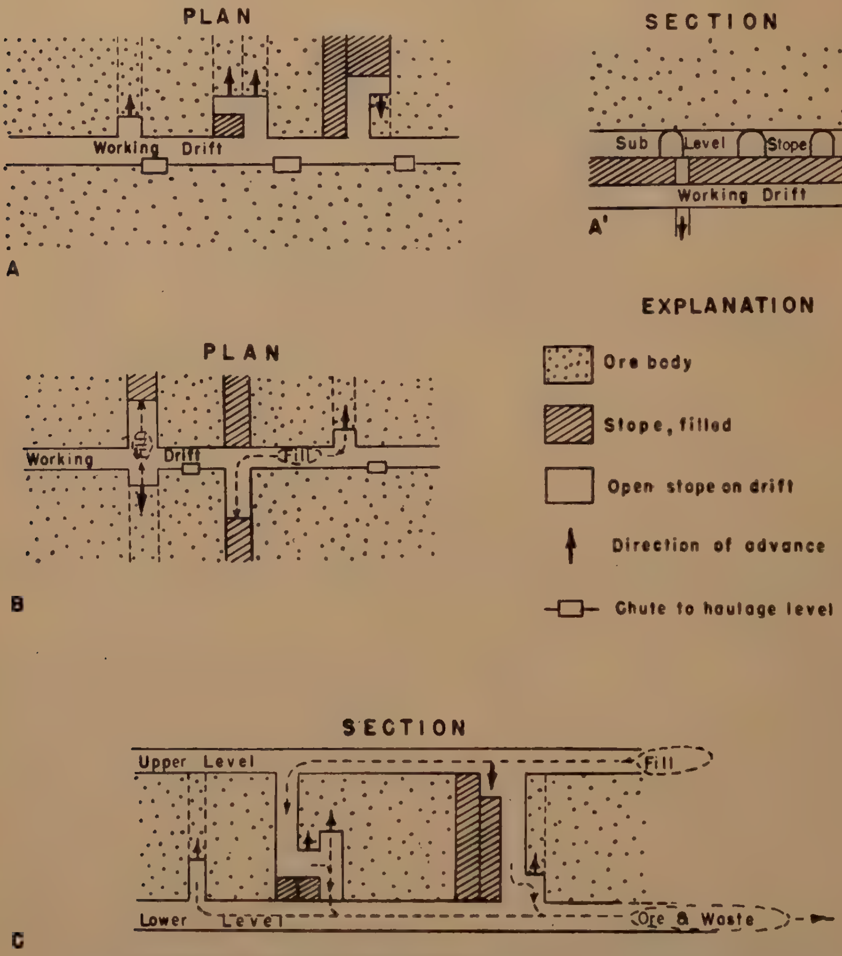

Fig 6—Sketches showing stoping methods at Abbadia San Salvatore mine.
*A*, *A'*—Plan and section show method used in normal ground.
*B*—Plan shows method for abnormally heavy ground.
*C*—Section shows proposed vertical slice method.

Mining is normally done with compressed-air rock drills and blasting, but in absence of power for compressors the Abbadia mine was achieving a sizeable production by pick and shovel methods, supplemented in places by hand drilling and blasting. Electric locomotives were formerly used in main haulageways, but donkeys and men were being used for tramming in March 1945.

Parts of some mines, particularly the Siele and Argus, are in sound rock, which will stand indefinitely with little or no support. Ordinary cap and post timbering with pine, beech, oak or other local woods, is used in working stopes or where the ground is relatively good. By far the greater parts of all mines, however, are in ground so heavy that the workings must be lined with brick, wood blocks or stone. Brick is used for lining almost all shafts and raises and is also used for many drifts. Linings of wood blocks, or of 10 to 14 in. hand-dressed blocks of trachyte are superior to brick and are widely used.

The newest method of support is to intercalate 3 or 4 lateral rows of wood blocks in the normal brick or rock lining. The wood compresses first and thus takes up pressure without distortion and rupture of the lining. Even this kind of lining, however, must usually be replaced at 6 month intervals.

All workings except stopes are either circular or oval in section so as to distribute the essentially fluid pressure of the ground. Inclined raises are fitted with stone stairways, made of trachyte or massive limestone, and with wooden handrails. Portals are of masonry or of reinforced concrete.

## METALLURGY

*Furnaces and furnace capacity*—The Abbadia and Idria mines use tower or shaft furnaces (forni a tino) for treatment of coarse ore and Cermak-Spirek furnaces for fine ores. The other mines depend entirely on Cermak-Spirek furnaces.

The tower furnaces are simply hollow vertical chambers, similar to small blast furnaces in most respects. They are claimed to be efficient, are cheap to build, and are more economical than Spireks because they require repair only once in 20 years whereas the Spirek furnace must be repaired every 4 or 5 years.

The Cermak-Spirek furnace is similar to the Scott, formerly used extensively in the United States. It is described and illustrated by De Castro[4] and by Duschak and Schuette.[1]

The Siele company was considering the installation of several rotary furnaces, at the time of the writer's visit, but there is general satisfaction in Italy with the present furnace methods. Both tower and Spirek furnaces are evidently efficient. The burned ore contains very little mercury and losses in fumes are negligible. There is no noticeable odor of sulphur fumes around the furnaces and cases of mercury poisoning among the 600 to 800 workers at Abbadia are limited to about only 3 per year.

Stack losses at Abbadia are said by the management to be less than 50 flasks per year. The operators believe that with their abundant labor and with ore that contains a high percentage of fines either rotary or multiple hearth furnaces would be less economical than their present equipment.

The Siele mine has 2 Spirek furnaces of 24 tons each daily capacity and 5 of 16 ton daily capacity. The rated capacity is 128 tons per day, but in practice this is usually increased to about 150 tons per day. Recovery claimed is 93 pct.

The Argus mine has 3 Spirek furnaces of 12-ton capacity and 1 of 18-ton capacity. In practice, 50 to 60 tons of ore are treated per day.

The Cerreto Piano mine has 3 Spirek furnaces of 16 tons daily capacity, but only one of them is normally used. During the height of its production the Siele company treated ore hauled from its mine in the other two Cerreto Piano furnaces.

The Idria mine has 13 tower furnaces of 10 to 12 tons daily capacity, but only 7 of these are normally in use at any one time. It also has 2 Spirek furnaces of 50 tons capacity each and 2 of 14 tons capacity. One of the latter is specially equipped to treat low grade ore (0.35 pct Hg). Reported recovery at Idria is only 85 pct.

The Abbadia was the only mine in the Monte Amiata district whose furnaces were damaged at the time of the German withdrawal. It originally had 14 tower furnaces, each of 11 tons daily capacity and 7 Spirek furnaces of 40 to 50 tons capacity. The total rated capacity was thus 450 tons per day. All furnaces were destroyed by the Germans, but by the end of September 1944 the company had rebuilt its tower furnaces and 3 of its Spireks. A fourth Spirek was in process of construction in March 1945 and the company had plans to rebuild other furnaces by the end of the year so as to attain a productive capacity of 3000 flasks of mercury per month. Of the present rated capacity of 265 tons per day, or 7500 to 7800 tons per month, less than 200 tons per day, or 5500 tons per month were being actually used at the time of examination. Recovery claimed is 95 pct.

## Ore Treatment Practice

Treatment practice at the Abbadia mine is described below. That at the Siele mine differs only in that ore is crushed to about 2 cm diameter before being furnaced, that wet ore is dried in a batch-type drying oven, and that tower furnaces are not used.

The ore is not sorted, except by removal, during mining, of barren limestone and of nodules of cinnabar. The limestone is used for backfill and the cinnabar nodules are broken up by hand and mixed with the mine-run ore.

Ore normally goes directly from the mine to a coarse grizzly screen where it is reduced by hand to pass the 10 cm openings. The excess of mine production over furnace capacity goes temporarily to stockpile bins before this grizzly operation. It is partly dried by natural drainage and evaporation in the bins but can only be removed from them by pick and shovel methods almost as difficult as the original mining. From the grizzly the ore goes to a wood-fired rotary kiln, 1 by 6 meters (approximately 3 by 18 ft) in size, where it is dried at low heat from the mine-run moisture of 18 to 8 pct moisture, which is the maximum allowable for Spirek furnaces. The dried ore passes through a trommel screen which separates it into sizes greater and smaller than 3 cm. Actually the coarse ore consists of balls of cinnabar-rich clay with small pieces of almost barren rock. The coarse ore, which amounts to about one-third of the total, goes to the tower furnaces; the fine ore goes to the Spirek furnaces.

Furnaces were fired with fuel oil before the war, but they now use either wood or wood gas produced by a "Gasogen" apparatus for fuel. Wood gas is preferred, as it is more efficient, produces less soot than wood, and increases the capacity of a furnace far above its rated capacity. Thus a 24-ton furnace can handle 40 tons of ore per day if gas is used instead of wood.

The condenser system is much shorter than is common in American practice, and consists of 4 or more parallel rows of tile condensers of six 12-ft columns each. The top of the condensers is sprayed continuously with cold water. From the condensers the fumes go directly into flues that lead to the main stack. Condensers are washed down once a month and the flues are cleaned once a year. Soots from cleanups at the Abbadia and Siele mines are mixed with lime and worked on a hoeing machine similar to that at the New Idria mine in California except that it is not heated and has no hood. The Abbadia soots contain 20 to 30 kg of mercury per quintal, or 5 to 6 pct. At the Argus mine the soots are dried to cake form and burned in a small gas-fired rotary retort. This retort, about 16 in. in diameter and 8-ft long, operates on the batch principle. It discharges the fumes into a vertical 4-ft coil of 3-in. pipe, half submerged in a tank of water for cooling. A stream of water is introduced into the coil near its intake end, so that the final product is a mixture of liquid mercury and water. Soots from this retort are put back into the Spirek furnaces. The reasons for the somewhat unusual cooling system and its relative efficiency were not made clear to the writer.

At Siele, 80 pct of the mercury is precipitated in the condensers, 15 pct in the channels leading to the flue, and 5 pct in the flue itself.

The burned ore is ordinarily dumped direct from the furnace into a stream and is carried away but riffles are placed in the stream at intervals for a long distance below the plant. These riffles collect "nuggets" of unburned or partially burned cinnabar that have passed through the furnaces. Waste accumulated during the winter dry season is stacked in the usual dumps.

### SYNTHETIC CINNABAR MANUFACTURE

The Abbadia San Salvatore mine has a small plant for manufacture of synthetic cinnabar (vermillion). The plant capacity is much greater than the demand for the product, so that the plant is normally operated less than one month per year. Twenty-eight standard colors are produced, ranging from light orange to a dark purplish red. The product is sold directly by the mining company to consumers. The United States is one of the chief buyers.

The manufacturing process is very simple. Mercury, flowers of sulphur, soda ash and potash, in pre-determined quantities, are placed in a cast iron cylinder roughly 8 in. in diameter, and 18-in. long, closed, and revolved at a moderately high rate of speed. The reaction time ranges from 7 to 20 min, but in order to ensure completion of the reaction, it is customary to revolve the mixture for 1 hr to 1 hr and 20 min.

The resultant cinnabar is washed 3 times in water, filtered, and dried on shallow trays in a drying oven. It is then standardized for color and packed in boxes (Europe) or corrugated cans (U.S.) for shipping.

Inasmuch as the soda ash and potash are completely removed during the washing process the final product is pure mercury sulphide, regardless of its color. According to the operators, the final color depends on temperature, time of treatment, and relative proportion of soda ash and potash. Thus, lighter-colored cinnabar is produced in winter than in summer, other things being equal. Also the more potash the lighter the color, and the shorter the time of treatment the lighter the color. The actual difference in color seems to be one of particle size, with the smaller sizes being lighter color than the coarser ones. If this is true, presumably the desired color range could be obtained by treating all batches alike, then grinding the product to desired fineness and sizing for color, but this method is not used at Abbadia.

### ECONOMIC FACTORS

*Power supply and requirements*—Total power requirements of the three main Monte Amiata mines total about 800 kw. The power supply available to the mines in March 1945, was only about 100 kw.

The Abbadia mine has its own hydroelectric power plant, which normally generates 300 kw at 4000 volts pressure but can step up to 400 kw in winter. During the lowest recorded dry season the production was 166,000 kw-hr a month but even then the company had to buy only 34,000 kw-hr. The plant was undamaged by the Germans, but all power beyond a minimum required for maintenance was allocated to other users. The normal requirements are 100 to 140 kw for ventilation, 70 kw-hr for pumping, and most of the remaining power for hoisting, compressing, and the like.

The Siele mine formerly had its own hydro-electric plant but this was entirely destroyed by the Germans. The mine normally requires 400 kw for operation or 200 kw for maintenance. In early 1945 it was supplied with 80 kw by a Diesel generator; this was sufficient for pumping but did not allow for ventilation.

The Argus mine normally buys 60,000 to 80,000 kw-hr a month for operations and needs about 55 kw for maintenance. Its only source of power in 1945 was a small Diesel plant which supplied enough for repair of the shaft but did not permit ventilation or maintenance of the main workings.

*Labor*—The Abbadia mine normally employs 650 to 700 men, but had 850 men on the payroll at the time of the writer's visit. This increase was because of a "make-work" or relief program on the part of the company, the additional men being employed at various tasks on the surface.

Before the war, wages were 32 lire per day, "plus 60% for taxes, insurance and the like," or a total of 51 lire per day. Wages in March 1945 were 180 lire per day, "plus 60% for taxes, insurance, etc.," or a total of 288 lire per day. Furnace men work 6 hr; all others work 8-hr shifts. Wood cutters are paid by weight or volume rather than in day wages. Labor is decidedly restless, with strong Communist leanings, and is making periodic demands for higher pay. The company maintains a hospital and formerly ran company stores, but cooperative stores have now been established. There is considerable poverty and disease, especially tuberculosis, in the mining town of Abbadia San Salvatore. The Government old-age pension was 5 lire per day at a time when rationed bread sold for 12 lire per loaf. The company maintains excellent quarters for its staff and was building a few houses, away from the old town, for some of the laborers.

The Siele mine normally employs 450 to 500 men. In 1940 to 1941, during the height of its production, it employed 750 to 800 men; in 1942 to 1943 it had 550 to 650, and in March 1945 it had about 525, with a staff of 36. As the mine and plant were not operating at that time, it is obvious that much of the present employment was in the form of relief work.

The Argus mine normally employs 300 men, of whom 130 are underground workers. In March 1945 it had only about 25 men on maintenance work. The Cerreto Piano mine normally employs 110 men.

Under conditions that obtained in early 1945, with low furnace capacity, lack of supplies and machinery, and swollen employment, the Abbadia mine was producing about 1.7 flasks of mercury per man month. Before and during the war both the Abbadia and Siele mines were producing at more than twice this rate per unit labor.

*Cost of production*—The mine operators were naturally reluctant to give information on costs of production but it was possible to estimate the costs fairly well on the basis of labor costs and other known factors. In view of the generally held American belief that low foreign labor costs mean low production costs, it may be rather surprising to learn that the estimated production costs per flask of mercury are not greatly below those of several mines in the United States. This is true even without making allowance for the many million lire needed to repair and replace war-wrecked equipment.

## SELF-SUFFICIENCY OF ITALIAN MINES

The mines of the Monte Amiata district are remarkably independent of outside supplies or assistance as compared with most American mines. The labor supply is stable, for the entire population of Abbadia San Salvatore (population, 7000) as well as of Pian Castagniaio and several smaller villages, depends upon the mines and on tiny hillside farms for livelihood. The companies own their own timber lands and thus produce all their fuel (wood or wood-gas) as well as mine timbers and other timber. The mines cooperate with the Government in an active reforestation program so as to maintain the timber supply. Tile and common brick are made locally, and so is lime. Refractory brick was formerly brought in from outside, but it is now made locally of clay or rock. Intricate refractories for Spirek furnaces were being hand-carved from trachyte at the time of the writer's visit. Building stone, both common and ornamental, is quarried locally. Domestic and industrial water supplies are abundant and in normal times the mining companies generate nearly all of their own power. Materials are transported locally in carts drawn by oxen, mules, men or children. All the mines have well equipped machine shops as well as excellently trained technicians and engineers.

In addition to power, minimum needs for operation seem to consist of explosives, carbide, flasks, and means of transportation to and from the district. It is obvious, of course, that continuous operation would call for a continuous outside source of food, machinery, rails, cement, and many other items.

### ACKNOWLEDGMENT

Hearty acknowledgment is due the entire staff of the Mining Division, Allied Commission, who put their entire facilities and all of their data at the writer's disposal. Special thanks are due to Captain Donald S. Kingery, Corps of Engineers, Chief of the Division, Captain A. F. Radcliffe, Royal Engineers, in charge of the division's mercury investigations, and Corporal M. S. Mele, who acted as interpreter in the field.

Acknowledgment is equally due the managements and staffs of the three largest mines in the Monte Amiata district, who generously gave of their time, knowledge and hospitality. Those deserving special mention are Ingegneri Umberto Cattania, Giovanni Masobello and Ernesto Contini, respectively Consulting Engineer, Technical Director and Mine Superintendent of the Monte Amiata Company, Signor Giovanni Montagna, Administrative Director of the same company, and Ingegnere Carlos Spirek of the Siele Company. Without the sincere and efficient cooperation of all the named individuals, as well as of many others, the investigation could not have been successful.

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## DISCUSSION

C. N. SCHUETTE*—Mr. Eckel's paper is most interesting and timely. Speaking of the Monte Amiata District (p. 288), he says: "The district contains several smaller mines, some of them with sizeable reserves, as well as several promising prospects; but as long as the ore reserves of the principal mines remain so large and rich there is little incentive for opening them or for further search for new ones." Here is a great economic precept, succinctly stated. It holds true, not only for the Monte Amiata District, but for the world's quicksilver industry as a whole. This precept is the underlying reason for the ups and downs of our domestic quicksilver industry. When we have unhindered access to metal from the large and rich mines of Italy and Spain, our industry languishes from lack of incentive. At the present writing (March 1948) one large world source of quicksilver, the Idria District, is beyond the "iron curtain" and the Italian mines are in danger of a similar fate. Even Spain would not be too secure if Italy succumbed to red totalitarianism. It may behoove us to keep a mobile quicksilver industry alive behind our Maginot Line of a quicksilver stockpile.

Further along, Mr. Eckel mentions the dearth of recent geological reports and the fact that the latest one stresses lithology almost to the exclusion of structural geology. In going back through geological literature this change of emphasis from structure to rock classification can be noticed around the turn of the century when petrography grew out of petrology. Could it be due to the fact that examining thin sections in a laboratory is easier than climbing through dirty mine workings? It may be easier but definite instances of finding ore by this means are rare indeed. On the other hand many a miner has taken a lease on abandoned parts of a mine in which he had been working and has found ore. I know of many such cases where the leaser's knowledge of petrology was practically nil but where his horse-sense interpretation of geological structure brought tangible results. It would not surprise me to learn that most of the ore bodies in the Italian quicksilver mines had been found by the men actually working underground who may not write papers about it, but do find and follow the ore by applying the know-how accumulated through years of experience in their particular mines. It would be of interest to know how exploration and prospecting are carried on, and the fact that the author says nothing on the subject lends weight to the idea that ore discoveries are considered to be the routine job of the mine crew rather than achievements of geological prognostication by the corps of engineers and managers.

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Throughout the paper one is struck by the similarities and parallels with California quicksilver districts. For instance, Mr. Eckel states that ore has been found in all but the Quaternary rocks and that the age of mineralization is of late or post Pliocene age. That is true also in California. The section entitled: "Springs and Gaseous Emanations" (p. 291) could, with very slight modification, apply as well to the Mayacmas District in California. Apparently the smaller mines have an intermittent life similar to our own mines as Mr. Eckel says (p. 296 under "Reserves"): "The Argus workings and most of the smaller mines were inaccessible because of gas or caving . . . " Under "Mines" (p. 296) he says: "The discovery must have been one of nearly pure chance, for almost the entire surface area of the Abbadia deposit is hidden by unmineralized trachyte. This fact, as well as the presence of mineralized springs in the trachyte, lends hope for the ultimate discovery of other mercury deposits beneath the trachyte." In California we have similar possibilities of discovering additional quicksilver deposits under the Mark West andesite, under the trachyte flows at the Stayton mine and under other volcanic flows in the quicksilver districts, on the edges of which, quicksilver has been mined for many years.

The dearth of recent geological studies is regrettable but at least Mr. Eckel confirms one fact that I held to be significant, namely, that the base of the trachyte, which overlies the Abbadia deposit, is everywhere marked by 0.5 to 5 meters of black clay gouge. His east-west section in Fig 3, certainly pictures a mineralized zone capped by an impervious gouge. His Fig 2, of the Solforate del Siele mine also shows a flat-lying ore disposition but it is not clear from either the figure or from the text whether or not these ore bodies are capped by gouges or less pervious rock strata. The older geological papers which were available to me when I wrote "Occurrence of Quicksilver Orebodies," (ref. 3 of the paper) definitely tended to show that the Italian quicksilver deposits were formed by a primary concentration of cinnabar in trap structures.

In both mining and metallurgical practice the Italian quicksilver industry is still comparable to our pre-World-War-I practice to judge by the large number of men employed and the type of furnaces used. They are good furnaces and our own treatment costs per ton, when we were still using the Scott furnace, were lower than they are now with our mechanical furnaces.

Comparison of wages is a thankless job, particularly in times of fluctuating value of the currencies involved. But if the Italian cost per flask is not greatly below that of some of our mines, and I readily believe this, then the operating efficiency of our industry must be high indeed in view of our low-grade ores and the high pay and generally high standard of living of our workmen. This is also brought out by the following production figures:

| Country | Years Inclusive | Production in Flasks | Average Price, Dollars | Price Increase Per Cent | Production Increase, Per Cent |
|---|---|---|---|---|---|
| United States | 1935–1939 | 87,219 | 84.48 | | |
| | 1940–1944 | 222,773 | 177.43 | 110 | 156 |
| Italy | 1935–1939 | 271,788 | 69.98 | | |
| | 1940–1944 | 345,820 | 237.21 | 238(?) | 27 |
| Spain | 1935–1939 | 184,661 | 69.98 | | |
| | 1940–1944 | 293,080 | 237.21 | 238 | 59 |

Mr. Eckel does not state what price was received for the Italian production in the war years of 1940 to 1944, but no doubt it was comparable to the London or world price as it is not of record that a price ceiling was imposed on anyone other than the American producer. Our own industry, with a price increase of 110 pct from pre-war to war years, increased its production 156 pct. Spain, with a 238 pct price increase, upped its production by 59 pct, while Italian production was up only 27 pct with an unknown but probably commensurate price increase. Altogether this is not a bad showing for our "have-not" nation whose quicksilver resources were said to be practically exhausted. Perhaps the above example of results obtained by government controlled and nationalized mines vs private enterprise mines will give a hint to those who urge nationalization.

In conclusion, Mr. Eckel's paper gives us good factual data on which we can appraise the potential of Italian quicksilver production and he is to be congratulated on obtaining so much pertinent material in so short a time and under no doubt trying circumstances.

# Investigations of Mercury Deposits

By McHenry Mosier,* Member A.I.M.E.

(New York Meeting, February 1944)

## Summary

Mercury is one of the strategic metals of which the supply has been raised from critical uncertainty to more than enough for essential demands. Work by the Bureau of Mines has contributed substantially to increased production of mercury in the United States and to consequent accumulation of Government stocks. Other contributing factors have been higher prices for mercury and Government loans to operators.

The Bureau has undertaken 13 exploration projects, 10 of which have been completed. This work has comprised 28,376 ft. of several types of drilling and 23,798 lin. ft. of surface trenching, and has indicated commercial ore containing 34,800 flasks of mercury and marginal ore containing 26,200 flasks, a total of 61,000 flasks; equivalent, respectively, to domestic production for 8 months, 6 months, and 14 months.

As Government stocks are now adequate, domestic production (currently at the rate of 50,000 flasks per year) is gradually being curtailed through Government control of labor and operating supplies.

## Introduction

Mercury was one of the seven metals designated as strategic by the Army and Navy Munitions Board in 1939, when Congress appropriated funds for the investigation of domestic sources of supply of war minerals. The Bureau of Mines immediately started a limited search for strategic minerals, but the first mercury project was not begun until December 1940.

The U. S. Geological Survey assisted in the search for cinnabar by contributing information on domestic mercury deposits, preparing maps, and making geological studies of exploration projects. The present high rate of production of mercury in the United States was encouraged by the increase in price to $196 per flask at New York (Fig. 1) and Government loans to operators for mine development and plant construction.

The Bureau of Mines has issued, for restricted distribution subject to wartime controls, 13 War Minerals Reports on individual mercury deposits (listed in appendix A). In March 1942 it also published a Progress Report of the Metallurgical Division, *R. I.* 3627, Cinnabar-reduction Plants of the Southwestern Arkansas Quicksilver District.

## Uses of Mercury

Mercury and its compounds have more than 3000 uses. Its principal use is as a catalyst in the manufacture of gases for chemical warfare; the next largest is in pharmaceuticals. Mercury is essential also for: (1) certain electrical apparatus, (2) industrial control instruments at many war plants, (3) munitions, including blasting caps, (4) insecticides in agriculture, (5) antifouling paint for ship bottoms, (6) dental preparations, (7) general laboratory use, and (8) amalgamation of free-milling gold ores.

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Manuscript received at the office of the Institute Nov. 16, 1943. Issued as TP 1697 in Mining Technology, March 1944.

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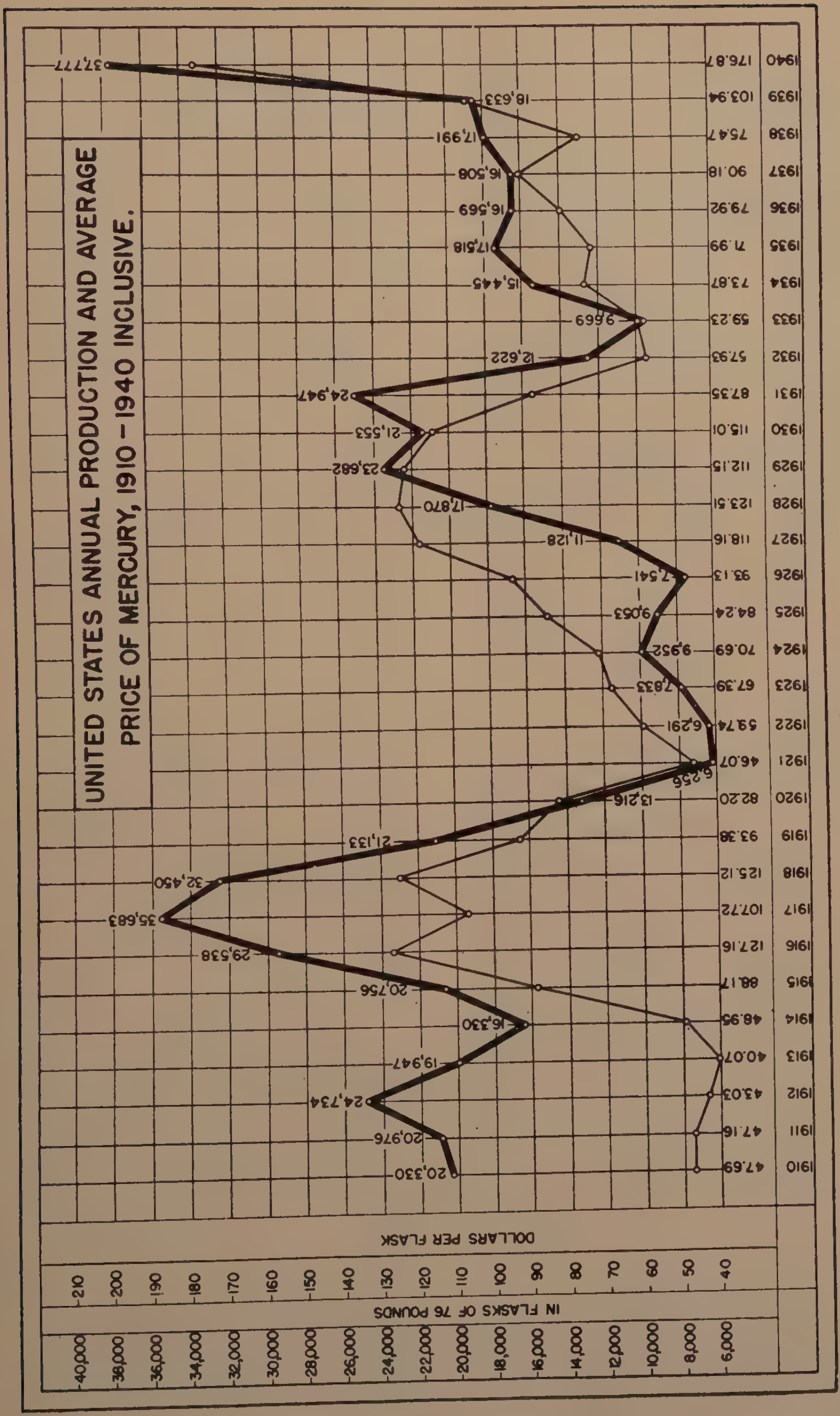
UNITED STATES ANNUAL PRODUCTION AND AVERAGE
PRICE OF MERCURY, 1910 – 1940 INCLUSIVE.
IN FLASKS OF 76 POUNDS
40,000
38,000
36,000
34,000
32,000
30,000
28,000
26,000
24,000
22,000
20,000
18,000
16,000
14,000
12,000
10,000
8,000
6,000
DOLLARS PER FLASK
210
200
190
180
170
160
150
140
130
120
110
100
90
80
70
60
50
40
1910 47.69 20,330
1911 47.16 20,976
1912 43.03 24,734
1913 40.07 19,947
1914 48.95 16,330
1915 88.17 20,756
1916 127.16 29,538
1917 107.72 35,683
1918 125.12 32,450
1919 93.38 21,133
1920 82.20 13,216
1921 46.07 6,256
1922 59.74 6,291
1923 67.39 7,833
1924 70.69 9,952
1925 84.24 9,053
1926 93.13 7,541
1927 118.16 11,128
1928 123.51 17,870
1929 112.15 23,682
1930 115.01 21,553
1931 87.35 24,947
1932 57.93 12,622
1933 59.23 9,669
1934 73.87 15,445
1935 71.99 17,518
1936 79.92 16,569
1937 90.18 16,508
1938 75.47 17,991
1939 103.94 18,633
1940 176.87 37,777

Fig. 1.

The Office of Production Management issued Conservation Order M-78 on Jan. 23, 1942, restricting or prohibiting the use of mercury for certain civilian uses, in order to conserve it for war purposes. Owing to improvement in supply compared with essential requirements in the United States, this Conservation Order was revised Sept. 9, 1943, to permit the use of mercury for some of the purposes formerly forbidden; but it restricted use to a quantity based on the amount consumed in a certain period in 1940 or 1941, as below: Marine antifouling paint, preparations for developing films, treatment of green lumber, wall switches for nonindustrial use, and wood preservation, 100 per cent; thermometers (except industrial and scientific) and vermilion, 80 per cent. The amount of mercury permitted in mercuric fulminate for commercial blasting caps was increased from 125 to 200 per cent. The amount allowed in industrial and scientific thermometers was revised from 100 to 200 per cent.

The amended order also provided that the restrictions of the Conservation Order did not apply to the consumption of mercury in: (1) medicinal or pharmaceutical preparations, including those for the treatment of livestock or poultry; (2) disinfectants, insecticides, fungicides (except turf fungicides) for agricultural use; (3) dental preparations; (4) medical, surgical, and dental equipment, instruments, and supplies; (5) X-ray apparatus and tubes; (6) laboratory instruments and equipment; (7) engineering instruments and equipment; (8) fire-protection equipment; (9) protective alarm and signal systems and industrial and safety equipment.

## OBJECTIVE OF EXPLORATION

The Bureau of Mines performs exploration principally (1) to increase ore reserves by delineating known mineral deposits, (2) to find additional ore bodies along mineralized zones that have proved productive, and (3) to determine subsurface ore horizons and structures containing ore. This last information would serve to control possible underground exploration and development by private industry. The Bureau is particularly interested in deposits that may contain large quantities of metals, including lean ores and those of marginal grade, but does not undertake primary prospecting or regional search for minerals. The ultimate purpose of the Bureau's exploration is to increase production of metals for war.

## PROCEDURE IN EXPLORATION

Preliminary information concerning mineral deposits is obtained in numerous ways. Many deposits are called to the Bureau's attention by other Government agencies, some by property owners or mine operators, and some by its own staff. Those that appear to have merit in connection with the Bureau's war program are examined by one of its engineers. (See appendix B.)

Some mercury deposits are so erratic that it is virtually impossible to block out ore in advance of its actual extraction. Such deposits generally can be explored only as they are mined. At deposits where exploration seems warranted, relative priorities are assigned and work is begun (in that sequence) as soon as funds, manpower, and equipment are available.

Although common methods of exploration and some innovations are employed, the chief reliance is upon diamond drilling and surface trenching. Although diamond drilling cannot be utilized fully at every mineral deposit, it has proved satisfactory wherever it has been tried by the Bureau. The objective is to determine subsurface conditions and to indicate ore rather than to block out measurable reserves. This method is flexible, speedy, and economical. Early in the exploration program the smallest standard bit (EX, ⅞-in. core) was used generally but later on the next size (AX, 1⅜-in. core) also was employed to obtain better core recovery and larger samples.

Contracts are let to individuals and firms in private industry for diamond drilling, churn drilling, and bucket drilling, either through competitive bidding or by negotiation. This work is supervised and sampled by Bureau engineers. Where bulldozers can be used for surface trenching, equipment and operator are contracted on an hourly basis.

The Bureau has not extended its mining operations beyond exploration. It does not develop or prepare mines for production nor engage in mining ores, except to obtain bulk samples for metallurgical tests. The only Government agency authorized to furnish funds to operators for development and mining is the Reconstruction Finance Corporation and its subsidiaries.

COST OF DIAMOND DRILLING

The Bureau of Mines paid contractors $3.84 per foot for diamond-drilling 207,817 ft. during the fiscal year ended June 30, 1943, at all exploration projects, regardless of metals sought or diameter of holes. Most of the footage was EX and AX. This was the basic contract price plus all extras, such as reaming, casing, cementing, and surveying. The total cost of drilling, including engineering, supervision, and sampling but not chemical analyses of the samples, was about $4.85 per foot drilled.

TABLE 1.—*Summary of Exploration for Mercury and Ore Reserves Indicated to Sept. 30, 1943*

| Project No. | Drilling, Ft.[a] | Underground Development, Ft. | Surface Trenches, Lin. Ft. | Indicated Ore Reserves | | | |
|---|---|---|---|---|---|---|---|
| | | | | Commercial | | Marginal | |
| | | | | Tons | Hg per Ton, Lb. | Tons | Hg per Ton, Lb. |
| 401 | 3,784[b] | | | | | | |
| 402 | | 28 | 8,193 | 2,200 | 10.5 | | |
| 403 | 3,539 | | 5,309 | | | | |
| 405 | 9,118 | | | 181,000 | 7.5 | 560,000 | 2.5 |
| 406 | 2,580 | | | | | | |
| 407 | | 58 | 240 | | | 171,000 | 2.0 |
| 408 | 2,496 | | | 76,000 | 3.0 | | |
| 410 | 3,090 | | | | | | |
| 412 | 1,500[c] | | | 4,000 | 10.0 | 164,000[d] | 1.5 |
| 414 | | 229 | 10,056 | 28,400 | 26.7 | | |
| 416 | | | | 56,500[d] | 2.3 | | |
| 416 | 1,000 | | | 34,500 | 2.2 | | |
| 420 | 1,269 | | | | | | |
| 421 | | | | 225 | 160.0 | | |
| Total | 28,376 | 315 | 23,798 | 382,825 | 6.9 | 895,000 | 2.2 |
| Flasks mercury | | | | 34,800 | | 26,200 | |
| Total flasks mercury | | | | 61,000 | | | |

[a] Diamond drilling, unless otherwise specified.
[b] Bucket drilling.
[c] Churn drilling.
[d] Mine dumps.

SUMMARY OF WORK PERFORMED AND ORE RESERVES INDICATED

Table 1 gives a summary of exploration for mercury by the Bureau of Mines and ore reserves indicated to Sept. 30, 1943. (For definitions of classes of ore, see appendix C.)

METALLURGY

Mercury ores from six deposits have been tested in this investigation, in addition to ores from exploration projects, to determine the most suitable method of treatment. Generally, either direct furnacing or pretreatment by flotation was applicable.

The standard metallurgical practice for ordinary ores of mercury is so simple that ore dressing (except crushing and screening) in advance of furnacing seldom proves profitable at modern plants. A concentrator is not required at any large mercury mine. Only for complex or refractory ores are such processes as flotation and hydraulic classification useful in connection with the discarding of finely divided particles of clay.

Almost all samples taken at exploration projects are sent to the Bureau's laboratories for assay and chemical analyses.

### Results at Individual Projects

All Bureau of Mines exploration projects for mercury and the results obtained from them to Sept. 30, 1943, are described in the following pages.

*Coso Deposit, Inyo County, California, Project* 401.—The Coso property was bucket-drilled, trenched, and sampled early in 1941, and 113,000 tons of material was indicated to contain 2 lb. of mercury per ton. At this project, 3784 ft. of hole 16 in. in diameter[*] was drilled with a rotary bucket. This is believed to be the first application of a "cesspool digger" to prospecting for mercury.

This submarginal deposit is a potential reserve for future exploitation should a national emergency require it.

*Bottle Creek District, Humboldt County, Nevada, Project* 402.—During the past four years, 5000 flasks of mercury have been produced in the Bottle Creek district. About 80 per cent was mined from ore bodies within or adjacent to diabase dikes. In 1940 and 1941, prospecting consisting of 6210 lin. ft. of trenching with bulldozers, 1983 lin. ft. of hand trenching, and 28 ft. of shaft sinking indicated 2200 tons of ore containing 10.5 lb. of mercury per ton. As all this ore has now been mined, the district has virtually no commercial reserves, although further prospecting may develop additional ore.

A magnetometer survey of only 10 acres by the Geophysical Section of the Bureau of Mines in February 1943 indicated three diabase dikes in addition to the 14 previously known in the district.

*Pike County, Arkansas, Project* 403.—Five properties in the western part of the quicksilver-producing district of Pike County were explored in 1941 as Project 403. Trenches aggregating 5309 lin. ft. were excavated, 12 holes aggregating 3539 ft. were diamond-drilled, and 267 samples of 100 lb. each were taken and assayed. A small quantity of submarginal material was discovered, but no commercial ore was indicated.

This work was done in only a small part of this large producing district, and further exploration may discover additional ore deposits.

*Hermes Mine, Valley County, Idaho, Project* 405.—Although the Hermes deposit was found in 1917, regular production did not begin until early in 1942, when a rotary furnace was installed on the property. At that time little was known of the grade of the ore reserves or of the potential output of mercury. The Bureau started diamond drilling in 1941, about the time that the Bonanza Mines, Inc., began to rehabilitate the mine. During 1941 and 1942 drilling of 9118 ft. increased the indicated ore reserves as shown in Table 2.

Table 2.—*Reserves Indicated by Drilling in* 1941 *and* 1942, *Pike County, Arkansas*

| Class of Ore | Tons | Hg per Ton, Lb. | Total Number Flasks Hg | Cut-off Hg per Ton, Lb. |
|---|---|---|---|---|
| Commercial.. | 181,000 | 7.5 | 17,800 | 4 |
| Marginal..... | 560,000 | 2.5 | 18,500 | 2 |
| Submarginal. | 231,000 | 1.1 | 3,400 | 1 |

Such substantial reserves justified installation of a second rotary furnace, which was undertaken in October 1942. About 140 tons of ore per day is being produced by the square-set method of stoping. The output of mercury is 400 flasks per month.

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[*] Includes 184 ft. of hole 26 in. in diameter.

The Bureau of Mines made ore-dressing tests to determine methods of handling the clayey ore found in some parts of the mine. These indicated that 35 to 40 per cent of the clayey material could be discarded by disintegration and hydraulic classification, with a loss of only 3 to 7 per cent of the mercury. The remaining product would then be suitable for dewatering and direct furnacing.

*New Idria Mine, San Benito County, California, Project* 406.—Sixteen diamond-drill holes, from both surface and underground, totaling 2580 ft., were drilled at the famous old New Idria mine from November 1941 to June 1942. This program developed information regarding geologic formations and structure that proved valuable to the operators in planning the underground prospecting that led to the discovery of high-grade cinnabar.

This was the most difficult of all ground to drill. Much of the drilling was in silicified sandstones and shales that yielded poor core recovery. Penetration was slow, in a few instances only 1 to 3 ft. per shift. Several holes had to be abandoned before their objectives could be reached. The acidity of the underground water, the presence of methane, and the tendency of the ground to cave easily made cementing impracticable in many holes.

The New Idria mine is the largest producer of mercury in the United States today, output being 1500 flasks per month. Square-set stoping is the standard system of mining.

*Silver Spots, Nevada and Oregon, Project* 407.—The object of Project 407 was to designate possible areas for exploration by the Bureau and to aid operators in maintaining current production. Twenty-five properties in Humboldt, Washoe, Esmeralda, Pershing, Nye, Elko, Mineral and Storey Counties, Nevada, and Malheur County, Oregon, were sampled to determine ore reserves and possible production. The work included 240 ft. of trenching, 28 ft. of test pitting, and 30 ft. of underground development and 4271 samples taken from November 1941 to August 1942. The principal properties sampled were:

| MINE | COUNTY | STATE |
|---|---|---|
| The Rand mine............ | Elko | Nev. |
| B. and B. mine............ | Esmeralda | Nev. |
| Pershing mine............. | Pershing | Nev. |
| Washington Hill........... | Storey | Nev. |
| Mercury Mining Co........ | Nye | Nev. |
| Last Chance............... | Humboldt | Nev. |
| Buckskin Peak............. | Humboldt | Nev. |
| Cordero mine.............. | Humboldt | Nev. |
| Bretz mine................ | Malheur | Ore. |

This work indicated 171,000 tons of marginal ore containing 2 lb. of mercury per ton. The Bureau's exploration project at the Bretz and Cordero mines resulted directly from the "silver spots" sampling.

*Black Butte Mine, Lane County, Oregon, Project* 408.—Diamond drilling at Black Butte mine, together with 900 samples taken from underground workings in 1942, has indicated 76,000 tons of ore reserves containing 3 lb. of mercury per ton. As the ore bodies were not completely delineated, additional ore may be developed. Seven holes totaling 2496 ft. were drilled.

*Terlingua District, Brewster County, Texas, Project* 410.—Project 410 was begun in June 1943, and 12 holes totaling 3090 ft. were drilled to Sept. 15, 1943. Diamond drilling continues in the Chisos and Fresno mine areas. Although no ore reserves have been indicated, two veins containing mercury were found on which underground development might be justified.

*Bretz and Cordero Mines, Humboldt County, Nevada, and Malheur County, Oregon, Project* 412.—As a result of joint drilling by the Cordero Mining Co. and the Bureau of Mines, completed in April 1943, about 4000 tons of ore reserves containing

10 lb. of mercury per ton was indicated at the Cordero mine. The Bureau did 1500 ft. of churn drilling. The operator continued drilling in areas outlined by the Bureau and has developed considerable additional ore.

Sampling of the mine dump at the Bretz property showed 164,000 tons of marginal material containing about 1.5 lb. of mercury per ton.

*Sleitmut Area, Sleitmut, Alaska, Project* 414.—Five properties in the Sleitmut area, Alaska, have been explored by 10,056 lin. ft. of surface trenching and 229 ft. of underground development. Two commercial and one marginal deposit have been found.

At the Red Devil deposit, 204 ft. of drifting and crosscutting found four lenses of ore with an indicated total of 11,000 tons containing 42.7 lb. of mercury per ton. Further exploration might disclose greater reserves.

The Alice-Bessie deposit, as determined by surface trenches and previous underground development with a 6-lb. cut-off on the 210-ft. level, contains 17,000 tons of ore averaging 16.5 of lb. mercury per ton. Inferred reserves are several times this amount.

Two deposits have been opened on the Barometer property by surface trenching and underground development. One has a small indicated reserve containing 11.6 lb. of mercury per ton, and the other contains 400 tons at 12.8 lb. Additional favorable ground remains to be prospected.

Good recoveries of antimony and mercury were obtained from a sample from the Sleitmut area by gravity, flotation, and retorting.

The New Idria Alaska Quicksilver Mining Co. is operating a 40-ton rotary furnace, which was installed in September 1943.

*Guadalupe Mine, Santa Clara County, California, Project* 416.—Mine dumps were sampled at the Guadalupe mine and trenching and diamond-drilling was done in promising areas. The larger dumps were trenched to bedrock by draglines. Several bulldozer cuts were excavated in surface material. From these exposures, 419 samples were taken. Favorable silica-carbonate rock was diamond-drilled—seven holes totaling 1000 ft. The project was completed in March 1943.

The mine dumps contain 56,500 tons averaging 2.26 lb. of mercury per ton. Surface material contains an indicated 34,500 tons containing 2.2 lb. per ton, which can be mined by opencut methods. The operators have sunk a shallow shaft near one of the Bureau's drill holes and have found an undetermined amount of high-grade ore. They are replacing an antiquated reduction plant with a rotary furnace.

*New Almaden Mine, Santa Clara County, California, Project* 420.—The New Almaden mine was the first producer of mercury in the United States. As known ore reserves have been almost exhausted, and as most of the richer mine dumps and stope fills have been worked, additional reserves must be found soon if production is to be maintained.

The Bureau of Mines started a diamond-drilling project at the New Almaden mine in August 1943 and by Sept. 30, 1943, had drilled 1269 ft. No ore was found, but geologic structures and contacts were indicated, together with a promising thickness of silica-carbonate rock, which is a favorable host for ore.

*Decoursey Mountain Mine, Flat, Alaska, Project* 421.—Project 421 was undertaken in June 1943. As the Decoursey Mountain mine is the least accessible of the projects in Alaska, the Bureau is assisting mine operators to complete an air field with a 1000-ft. landing strip.

Although nine veins were trenched by the operators, the tunnel ore body was the principal deposit found. This was sampled by the Bureau through a strike length of 210 ft. A preliminary estimate of ore reserves above the tunnel level is 225 tons containing 160 lb. of mercury per ton.

A bulldozer operated by the Bureau has trenched and stripped a portion of the Decoursey No. 1 ore body. In October this work was suspended until next spring.

The New Idria Alaska Quicksilver Mining Co. is operating two D-type retorts that were installed in September 1943.

## CONCLUSIONS

The Bureau of Mines is continuing exploration for mercury on a small scale, at five active projects, as a safeguard against any unforeseen sustained demand. Should necessity again arise for increasing the supply, this work, if successful in finding ore, will doubtless reduce the lag from exploration to production.

## APPENDIX A.—*War Minerals Reports Published by the Bureau of Mines as of September 30, 1943*

| SERIAL NUMBER | REPORT ON |
|---|---|
| | *California* |
| 14 | Guadalupe mine, Santa Clara County |
| 111* | Guadalupe mine, Santa Clara County |
| 116 | Coso mercury deposit, Inyo County |
| 121 | Falcon mine, Santa Barbara County |
| 203 | New Almaden mine, Santa Clara County |
| | *Idaho* |
| 106 | Bonanza (Hermes) mine, Valley County |
| | *Nevada and Oregon* |
| 46 | Bretz mine, Malheur County, Ore., Cordero Mine, Humboldt County, Nev. |
| 52 | Rand Mine, Elko County, Nev. |
| 48 | Black Butte mine, Lane County, Ore. |
| 112 | Nonpareil mercury mine, Douglas County, Ore. |
| 166 | Pacific syndicate mine, Jackson County, Ore. |
| 177 | War Eagle mine, Jackson County, Ore. |
| | *Texas* |
| 34 | Terlingua mercury district, Brewster County |

* Supplement to No. 14.

## APPENDIX B.—*Mercury Examinations by Bureau of Mines to September 30, 1943*

| LOCATION | NUMBER OF DEPOSITS EXAMINED |
|---|---|
| Alaska | 1 |
| Arizona | 7 |
| Arkansas | 29 |
| California | 34 |
| Colorado | 2 |
| Idaho | 16 |
| Michigan | 1 |
| Nevada | 37 |
| New Mexico | 1 |
| Oregon | 27 |
| South Dakota | 1 |
| Texas | 11 |
| Utah | 2 |
| Washington | 3 |
| West Virginia | 1 |
| Wyoming | 1 |
| | 174 |

## APPENDIX C

A new method for classifying ore reserves has been adopted by the Bureau of Mines and the Geological Survey.

*Measured ore* is ore for which tonnage is computed from dimensions revealed in outcrops, trenches, workings, and drill holes and for which the grade is computed from the results of detailed sampling. The sites for inspection, sampling, and measurements are so closely spaced and the geologic character is defined so well that the size, shape, and mineral content are well established. The computed tonnage and grade are judged to be accurate within limits, which are stated, and no such limit is judged to differ from the computed tonnage or grade by more than 20 per cent.

*Indicated ore* is ore for which tonnage and grade are computed partly from specific measurements, samples, or production data and partly from projection for a reasonable distance on geologic evidence. The sites available for inspection, measurement, and sampling are too widely or otherwise inappropriately spaced to outline the ore completely or to establish its grade throughout.

*Inferred ore* is ore for which quantitative estimates are based largely on broad knowledge of the geologic character of the deposit and for which there are few, if any, samples or measurements. The estimates are based on an assumed continuity or repetition for which there is geologic evidence; this evidence may include comparison with deposits of similar type. Bodies that are completely concealed may be included if there is specific geologic evidence of their presence. Estimates of inferred ore should include a statement of the spatial limits within which the inferred ore may lie.

# Colloidal Deposition of Cinnabar

By James Pollock,* Junior Member A.I.M.E.

(New York Meeting, February 1944)

## Summary

The possibility of colloidal deposition of cinnabar has been neglected. In opalite deposits cinnabar exists in particles within or near the colloidal size range. Colloidal processes have been admitted to be operative in hot springs and low-temperature deposits similar to most cinnabar deposits. Mercuric sulphide sols are stable to elevated temperature and pressure, and in nature probably are chiefly affected by the electrolytes in solution, dilution of colloid, and presence of protective colloids as silica. The stability of mercuric sulphide sols to electrolyte is measured, and shown to be greatly increased by dilution and by the protective action of silica. Mercuric sulphide sols follow the Schulze-Hardy valency rule, therefore the precipitating power of complex mixtures of electrolytes can be computed roughly. The stability of mercuric sulphide sols is shown to be greater than the equivalent precipitating power of the electrolytes in a hot spring depositing mercuric sulphide. It is concluded that finely divided or earthy types of cinnabar are colloidally deposited.

## Possibility of Collodial Deposition

While colloidal solution has been suggested as a means of transport for a number of minerals, such as the sulphides of antimony, arsenic, and iron, it has been neglected in connection with mercury. This is surprising because there exists positive evidence that cinnabar is present in certain deposits in particles of colloidal or near colloidal size.

Colloidal transport could result from either the rapid dilution or neutralization of alkaline sulphide solution bearing mercury coming in contact with deep ground waters, or from colloidal solution near the original source and subsequent flow to the point of deposition unchanged. That the dilution of a mercuric alkaline sulphide solution will precipitate mercury sulphide in colloidal form has been demonstrated a number of times and is easily confirmed. The dilution hydrolyzes the alkaline sulphide $Na_2S$ to the corresponding hydroxide and hydrosulphide NaOH and NaHS, and the diminution of the concentration of alkaline sulphide reduces the solubility.

It is not the contention of this writer that colloidal deposition is the mode of transport of mercuric sulphide in all mercury deposits, but that the stability of a mercuric sulphide sol, as demonstrated below, in waters that are similar to those that have deposited mercuric sulphide necessitates consideration of this possibility in many deposits, especially those of the hot-spring and so-called opalite type.

## Evidence of Colloidal Deposition

Ross[1] has shown that most deposits of quicksilver are formed at low temperatures and pressures, and probably by solutions that have relatively simple chemical compositions. In most cases the solutions are probably similar to those observed in hot springs, and in two instances—at Steamboat Springs, Nevada, and Sulphur Bank, California—hot springs have been reported by Becker as actually depositing cinnabar.[2]

Opalite deposits were believed by early investigators to be surface phenomena

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Manuscript received at the office of the Institute Nov. 16, 1943. Issued as TP 1735 in Mining Technology, July 1944.

* Ground Hog Mines, Vanadium, New Mexico.

[1] References are at the end of the paper.

similar to hot springs. This has been shown incorrect,[3] and it is probable that they are replacements at moderate depths by silica not necessarily opal. The similarity of opalite deposits and hot springs that led to the confusion of origins is striking in many respects. The mineralogy is very similar in regard to the chief constituents. The importance of colloidal processes in hot springs and low-temperature and pressure deposits has long been recognized, and this fact alone warrants review of the stability of mercuric sulphide sols in the environment, which may be visualized as that attained in the solutions that formed these low-temperature and pressure deposits.

The physical state of cinnabar deposited in the opalite type seems to the writer to offer direct evidence of colloidal deposition. The finely divided state of this cinnabar impresses all who have worked with this material. Yates[4] describes the cinnabar as "intimately mixed with silica in chalcedony." Attempts by Dreyer[5] to polish specimens of opalite-bearing cinnabar failed because of the small particle size. After some experiments in polishing briquetted, crushed, and sized opalite, he came to the conclusion that the cinnabar in opalite was very finely divided, considerably below 200 mesh.

Particles of cinnabar in specimens from the Opalite mine, McDermitt, Nev., examined in polished thin section by the writer could not be resolved by magnification of 600 diameters. The field of the lens used was 0.26 mm. at this magnification. One-hundredth part of this field could be distinguished, or 0.0026 mm. (2.6 microns). The upper limit of the colloidal range is given by Hauser (p. 25 of ref. 6) as 0.5 microns. Therefore in the opalite type of deposit particles of cinnabar exist in or nearly in the colloidal range.

The greater part of the cinnabar is present in similar finely divided condition or in particles that could be flocs or coagulated particles of colloidal cinnabar. A part of the cinnabar (p. 456 of ref. 1) is in thin films on fractures and apparently later than the disseminated variety. This material, however, is not in megascopic crystals. While megascopic and even large crystals are not unknown, the generally finely divided condition of the cinnabar contrasts with the type of precipitate; i.e., comparatively large crystals, obtained from alkaline sulphide solutions, even rather dilute solutions, in the laboratory when the experiment is carried on under geologic conditions of small and slowly added increments of precipitating agent.

When Becker visited Steamboat Springs, Nev., he collected a sample of the spring water, which he says[3] was depositing mercuric sulphide and formed a precipitate of arsenic and antimony sulphide and silica in the tube with which he siphoned the water from the spring. Becker believed that arsenic, antimony and mercury were present in this water as complex salts of sodium sulphide. This occurrence has been widely quoted as field evidence of alkaline sulphide transport of these metals.

However, the complex ions cannot have been present, for they become more soluble and not less soluble on cooling, which Becker states as being the sole agency effecting precipitation. This has been shown by Knox[7] and discussed by Dreyer[5] and Pollock.[8] Why Becker expected the metallic sulphide to precipitate from a cooled solution is not clear. Sodium mercuric sulphide is stable, does not break down upon evaporation, and is prepared easily by evaporating hydrous solutions,[7] therefore these metals cannot have been present in solution as the complex salt $Na_2XS_2$; if they had been they would have precipitated as the complex salts and not broken up to the simple sulphide.

As Becker states that $H_2S$ and $CO_2$ were being given off by the springs, and it has been shown that the metallic alkaline sulphide salts cannot exist in the presence of an excess of $H_2S$,[8] there seems no possibility that the complex salts can have been present. The assumption by Becker and others that this is field evidence of alkaline sulphide deposition is incorrect. Since the solubility of the metals in question in dilute alkaline solutions in the presence of $H_2S$ is very small, colloidal solution offers a possible explanation of their transport in the sample collected by Becker.

Consideration of colloidal deposition seems warranted by the similarity between opalite and hot-spring mercury deposits and hot springs where colloidal processes have been proved operative, and by the direct evidence of cinnabar in opalite in or near the colloidal range. This study is aided by the fact that there are available analyses of waters that were depositing or had deposited cinnabar recently.[2] Therefore we can study the stability of sols of cinnabar under conditions that probably are very similar to those attained in nature.

## Factors Affecting Stability of Colloidal Mercuric Sulphide

Mercuric sulphide and a number of other metallic sulphides, among them arsenic and antimony, have the property of being very easily peptized or dispersed into colloidal solution by $H_2S$. $H_2S$ bubbled through freshly precipitated mercuric sulphide produces a deep black sol in a short time. The dispersion and stability of the colloidal particles is caused by the adsorption of the negative $S^{=}$ ion and probably also the $HS^{-}$ ion. Therefore the charge of the particle should be negative. This was experimentally checked and found to be true.

The properties of a colloidal particle are the result of the charge on the particle, and the stability of the particle therefore is dependent upon the presence of the charge. If the charge is neutralized, the stability is lost, and the particle will flocculate with other particles. This neutralization can be accomplished by mixing sols of opposite charge, by mixing a sol with a source of ions of opposite charge such as an electrolyte, or by bringing the sol in contact with a source of charge of the opposite sign, such as a cathode or anode. For ore-bearing solutions the most important case is that of neutralization by electrolytes. Mutual coagulation of sols of opposite sign is probably rare in these solutions, as the sign of most metallic sulphide sols and that of silica is negative. The importance of the third possibility is in doubt, but at least in the stage of transport of the sulphides its action must be slight. However, the contact potentials developed by solutions of various types and various rocks do not seem to have been investigated, although electrodeposition is not only an important factor in the precipitation of sols but also in deposition of metals. Layers of rock of different composition in contact in solutions of electrolytes must act as weak galvanic cells.

In the case with which we are dealing, we may focus our attention on the chief cause of neutralization of charge and flocculation; i.e., adsorption of ions (here cations) of electrolytes. The power of ions to neutralize a charge varies with the valence. According to the Schulze-Hardy rule monovalent, bivalent, and trivalent ions have precipitation power roughly in the ratio 1:100:1000. Trivalent cations are not found as common constituents of hot-spring waters, and probably are rare in low-temperature ore solutions. In analyses given by Becker[2] of waters supposedly depositing cinnabar, the predominant ions at Steamboat Springs were $Na^{+}$, $K^{+}$, and $Cl^{-}$. At Sulphur Bank, California, they were $Na^{+}$, $BO_3^{-}$, $K^{+}$, $CO_3^{-}$, and $Cl^{-}$. At both localities $Mg^{++}$ and $Ca^{++}$ were present in minor amounts.

In addition to the valence, many other factors affect the precipitating power of the ion, chief of which are the physical state of the solution; i.e., temperature and pressure, dilution, and also the presence of ionic fields, both of the same charge and opposite charge, which may be caused by other ions (here cations) of the same charge and the ions (here anions) of the opposite charge, which are called stabilizing ions because they tend to neutralize the precipitating ion.

Mixtures of electrolytes, common in nature, have a variety of ionic fields caused by the various cations and anions. Complex mixtures have not been studied, but it is known that a mixture of two electrolytes of equal valence has precipitating power equal to the sum of the two alone. However, mixtures of electrolytes of different valences exhibit the phenomenon of ion antagonism. The mixing increases the concentration of electrolyte necessary to precipitate a given sol[9] and thus increases the stability of the sol. The magnitude of the effect in the range studied is about a 10 per cent increase in stability. While this might possibly be important in a solution approaching the critical point of stability, compared with the degree of stability experimentally found in mercuric sulphide sols, it is minor.

As ore solutions, whatever the temperature and pressure at the time of deposition, presumably have been subjected to higher temperatures, and certainly pressures at an earlier period in their history, the effect of these factors on the stability of sols is of interest. Von Buzagh[10] states that in general the stability of a sol increases with increasing temperature. Frondel[11] found an increase in the stability of gold sols on increase of temperature from 30° to 100°C. When heated in bombs, spontaneous coagulation of unprotected sols did not take place until a temperature of 150° to 200°C. was reached, and in sols protected by silica, not until a temperature of 400°C. was reached. In both of these experiments the pressure was also naturally increased. Unprotected sols of mercuric sulphide can be boiled for an extended period without coagulation, and can be considerably concentrated in this manner. Mercuric sulphide sols therefore are stable at 100°C., and undoubtedly at higher temperatures, especially in dilute solutions, and also when protected by silica.

In the application of results of work by colloid chemists to geologic problems, the concentration of the sol is seldom considered. Yet the chemist works habitually with concentrated sols, while the concentration of metallic elements in natural solutions is relatively low. In work seen by the writer, the concentration of colloid was never less than one gram per liter. Natural sols probably have much lower concentrations, as the ratio of silica to cinnabar in opalite deposits is at a minimum 500 to 1, judging from the grade of ore mined. In the water at Steamboat Springs, there was 0.31 gram of silica per liter in the sample taken by Becker.[2] This would give a mercuric sulphide concentration of 0.00062 gram per liter if the ratio of 500:1 held true. As shown by the experimental work described in the following paragraphs, such a dilute sol would have a stability to a given concentration of electrolyte many times that of a sol as concentrated as 1 gram per liter.

Although the Burton-Holmes so-called rule has been formulated concerning the behavior of diluted sols, actually there is considerable divergence between the rule and experimental results except for the sols with which Burton et al. worked.[12] As considerable changes in stability are involved, a brief review of the work of colloid chemists on the subject may be of interest.

Kruyt and Van der Spek,[13] the earliest workers, found that on dilution of an arsenic sulphide sol the precipitation value of KCl increased greatly (i.e., stability of the sol increased greatly), that of $BaCl_2$ decreased slightly, and that of $Al_2(SO_4)_3$ decreased rapidly. Similar results were obtained by Burton et al.[12] on a more extended scale on arsenic sulphide sols, and they formulated the Burton-Holmes rule:

1. "For univalent ions the concentration of ion necessary to produce coagulation increases with decreasing concentration of the colloid, this increase being very rapid for low concentrations of the colloid.

2. "For bivalent ions the concentration of the ion necessary to produce coagulation is almost constant and independent of the concentration of the colloid.

3. "For trivalent ions the concentration of ion necessary to produce coagulation varies almost directly with the colloid."

For trivalent ions above the precipitation value of the electrolyte decreased.

These results did not prove applicable to sols of other substances. Weiser and Nicholas[14] worked with dilution of sols of chromic oxide, prussian blue, and ferric oxide. In every case, contrary to the Burton-Holmes rule, the precipitation value of all electrolytes decreased upon dilution. However, Judd and Sorum[15] and Fisher and Sorum,[16] working with ferric oxide and chromic oxide sols, respectively, found that highly purified sols followed the Burton-Holmes rule, but that the addition of small amounts of impurities such as chromic chloride gave the results obtained by Weiser and Nicholas. Mercuric sulphide sols tested by the writer did not follow the Burton-Holmes rule and in every case dilution increased stability to electrolytes of all valences. This same departure from the Burton-Holmes rule has been observed by Mukherjee and Sen,[17] who say: "In the case of mercuric and cupric sulphide sols it is found that dilution increases the stability of the sol irrespective of the nature of electrolyte." Whether this departure from the Burton-Holmes rule is caused by impurities has apparently not been investigated and the function of the impurities in imparting large increases in stability has not been satisfactorily explained. It is apparent, however, that the so-called rule is not a reliable guide to the behavior of diluted sols, and that, contrary to this rule, considerable increases in stability have been found for mercuric sulphide sols upon dilution. The factor of dilution of the colloid therefore is of great importance in application of colloidal chemistry to geologic problems.

## Stability Measurements of Mercuric Sulphide Sols

It has been shown in the foregoing pages that the most important factors in the stability of sols from a geologic point of view are the temperature, pressure, dilution, electrolytes, and protective colloids present. In mercuric sulphide deposits, temperature and pressure are low, and hence assume a minor role, and detailed experiments were carried out only for the three factors last named.

The technique used in the measurement of stability has been described in detail.[19] It is customary in colloid chemistry to measure stability by determining the concentration of a given electrolyte that will not precipitate a sol in a given time, usually two hours. In order to conserve time, the writer instead has varied the concentration of electrolyte over a considerable range and measured the time in which precipitation or flocculation takes place. This method consumes much less time, and has the advantage that the shape of the precipitation time-concentration curve is shown, and hence the rate of change of precipitation time. The point on these curves at a time of two hours is of course equivalent to the customary precipitation value (i.e., concentration) of electrolyte. The precipitation time was defined as the time necessary for colloidal particles of mercuric sulphide in contact with the concentration of electrolyte in question to flocculate to such a size that they would not pass through No. 1 Whatman filter paper.

Sols used were as follows:

Sol X (HgS), concentration 1.8 grams per liter
Sol Z (HgS), concentration 2.0 grams per liter
Sol A ($SiO_2$), concentration 7.4 grams per liter

Sols X and Z were prepared from c. p. $HgCL_2$ by precipitation of HgS with $H_2S$, repeated washing with distilled water, and then dispersion of the HgS with $H_2S$. Sol A was prepared from c. p. sodium silicate and electrodialyzed one week.

Standard solutions of three electrolytes, NaCl, $CaCl_2$, and $AlCl_3$, and also NaOH and HCl, were made up and diluted to the concentration desired. Electrolytes in amounts of 1 c.c. were added to 10 c.c. of sols in a test tube. The test tube was shaken vigorously several times, and allowed to stand until the appearance suggested flocculation. The solution was then filtered. When no trace of color remained in the solution, flocculation was deemed complete. Once the approximate time was found, checks were run until the time was determined within about 20 sec. Duplicate runs were made on sol X and successive dilutions to ½, ¼, ⅛ the original concentration.

Precipitation values vary even in much more carefully controlled experiments because the stability of the particle is affected by the method of preparation, the particle size and age, the presence of precipitating or stabilizing impurities, the manner in which the electrolyte is added, etc. The writer's method is subject to several objections. First, there was no constant method of adding electrolyte nor of mixing. Second, filtration to determine the degree of flocculation removes the already flocculated sol and leaves a diluted sol. A diluted sol is more stable, and this contributes some error to the process. However, results are reproducible within the limits of error of 20 seconds.

### Stability to Electrolytes

The results of measuring precipitation time to various concentrations of electrolytes are plotted in Fig. 1. The concentration of electrolyte for the point at which the curves become approximately asymptotic to the abscissa (time axis) is equivalent to the customarily used precipitation value. In these curves, while the scale of time, the abscissa, is constant, the scale of the ordinate changes in the ratio of 1:100:1000 as does the precipitation value of the electrolytes (Schulze-Hardy rule). The precipitation value for NaCl in millimols per liter is approximately 50, for $CaCl_2$ is 1, for $AlCl_3$ is 0.08. The valency ratio, therefore, roughly follows the Schulze-Hardy rule, the relative efficiency as precipitants being 1:50:625. The difference is such that use of the Schulze-Hardy rule would exaggerate the efficiency as precipitants of bivalent and trivalent ions.

In NaOH, the stabilizing ion $OH^-$ has a marked effect on the precipitating power of the $Na^+$ ion as compared with the $Na^+$ ion in NaCl. The precipitation value is reduced from 50 to 30 millimols. The effect of the OH ion should stabilize the sol and raise the precipitation value instead of reducing it. As is to be expected, the $H^+$ ion of HCl has a marked precipitating effect on the electronegative sol, and hence a low precipitation value. Mercuric sulphide sols tend to be stable in alkaline solutions and unstable in acid ones.

### Stability to Dilution

As discussed above, the effect of dilution upon stability holds special interest because sols in nature may be expected to be far less concentrated than those with which the colloid chemist works. Sol Z was diluted to concentrations of 0.5 Z, 0.125 Z, 0.0625 Z, etc. The concentration of electrolyte was held constant, and 10-c.c. portions of the various dilutions of the sol were mixed with the 1-c.c. of electrolyte and flocculation time observed. In the run with NaCl, the concentration of electrolyte in the test tube after mixing was 0.388 molar, in the run

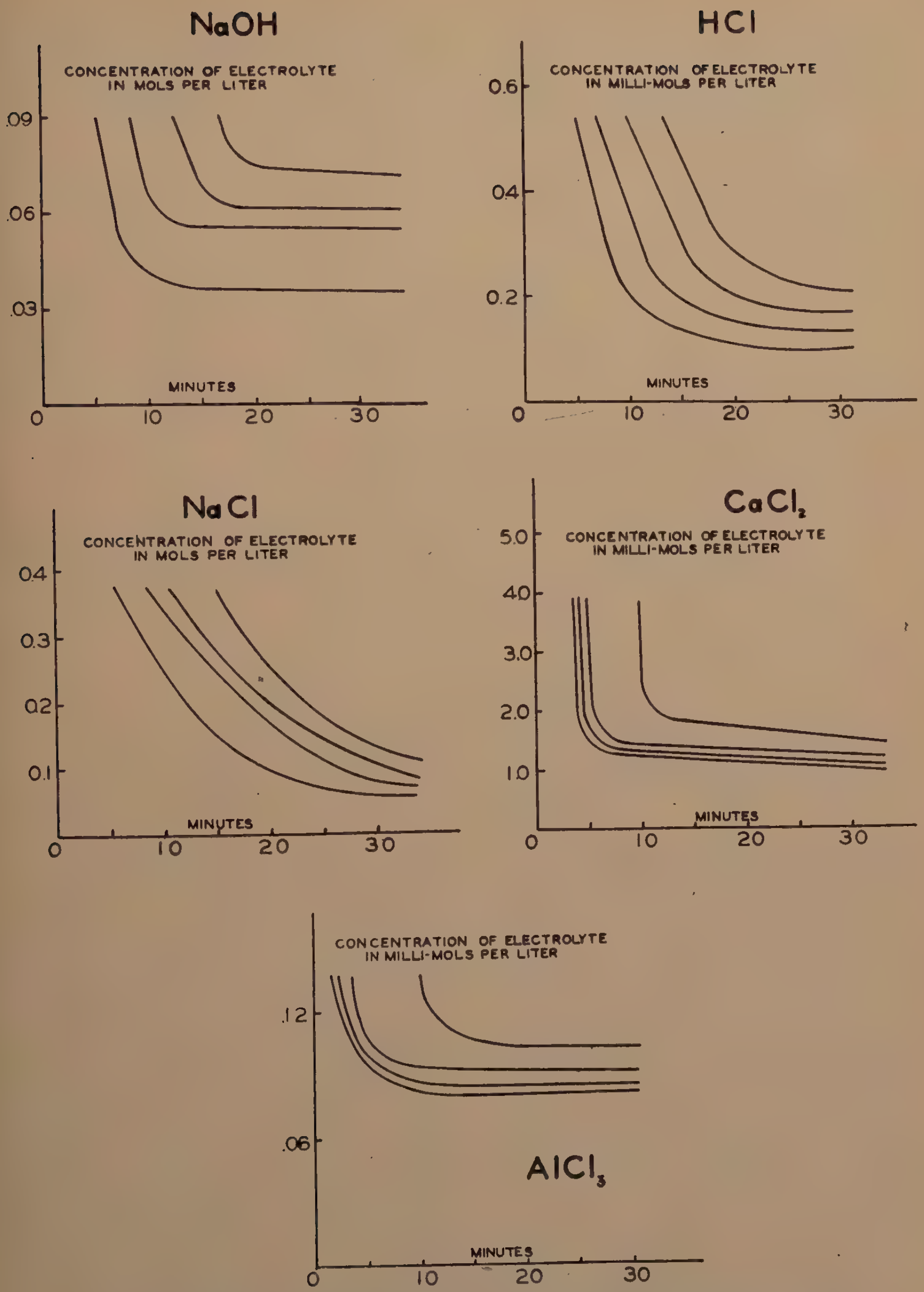

FIG. 1.—STABILITY OF HgS SOL TO VARIOUS ELECTROLYTES.
Curves shown for concentration of sol of 1.8, 0.9, 0.45, and 0.22 grams per liter.

with $CaCl_2$, 0.0035 molar, and with $AlCl_3$, 0.00013 molar.

The results are plotted in Fig. 2. In each electrolyte the stability of the sol (i.e., precipitation time) increases with dilution. The change takes place at a rather definite point. This shows that sols of mercuric sulphide do not follow the Burton-Holmes rule. The increase in stability is large, and effectively becomes infinite for the concentration of electrolytes used at a concentration of 0.06 grams per liter of mercuric sulphide. The factor responsible for such an increase in stability would appear to deserve more attention than it has received in the question of colloidal deposition.

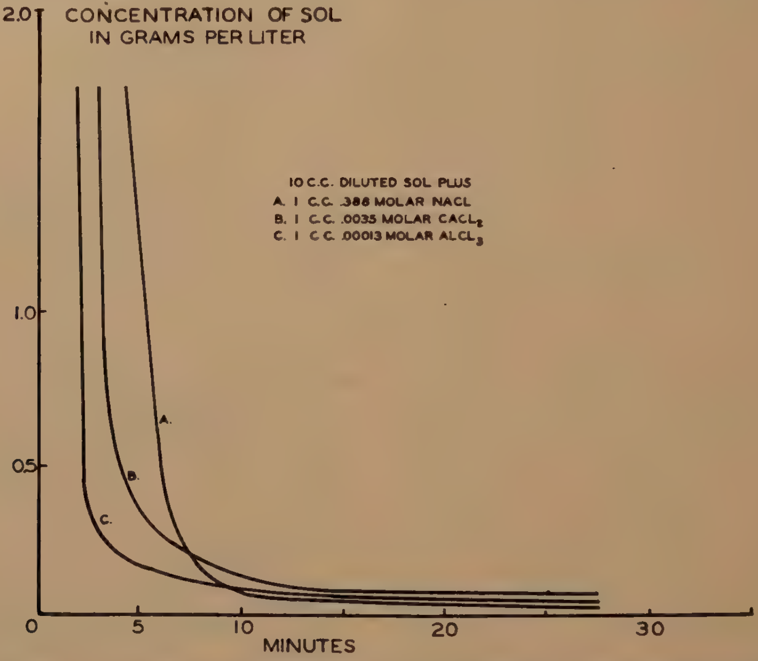

FIG. 2.—EFFECT OF VARYING CONCENTRATION OF SOL ON STABILITY, ELECTROLYTE CONCENTRATION BEING HELD CONSTANT.

### STABILITY TO PROTECTIVE COLLOIDS

The protective action of colloidal silica to many other colloids is too well known to require further comment. As shown in Fig. 3, silica exhibits that protective action toward mercuric sulphide sols. The degree of protection may be surprising. In Fig. 3, the precipitation time of 5 c.c. of sol A diluted to 1.0 gram per liter plus 5 c.c. of sol Z diluted to various silica-mercuric sulphide ratios is shown. The precipitant is 1 c.c. of 0.338 molar NaCl in each case. At a ratio of 3.7 to 1 of silica, the precipitating time become effectively infinite. In natural solutions, as discussed above, the ratio of silica to mercuric sulphide is much higher.

An explanation of this rapid and large increase in stability may be found in the protective mechanism. Protection is discussed by Hauser (p. 148 of ref. 6). When the particles of the protective colloid are large with respect to the protected colloid, the protected particles (here mercuric sulphide) are adsorbed by the protective particles, become immobilized, and adopt the properties of the protective agent (here silica). That is the mercuric sulphide sol when protected by a sufficient ratio of silica sol will become effectively a silica sol and loose the characteristic sensitivity to small amounts of electrolytes. When this happens, the question of colloidal transport becomes one of the stability of silica sols and not of mercuric sulphide sols.

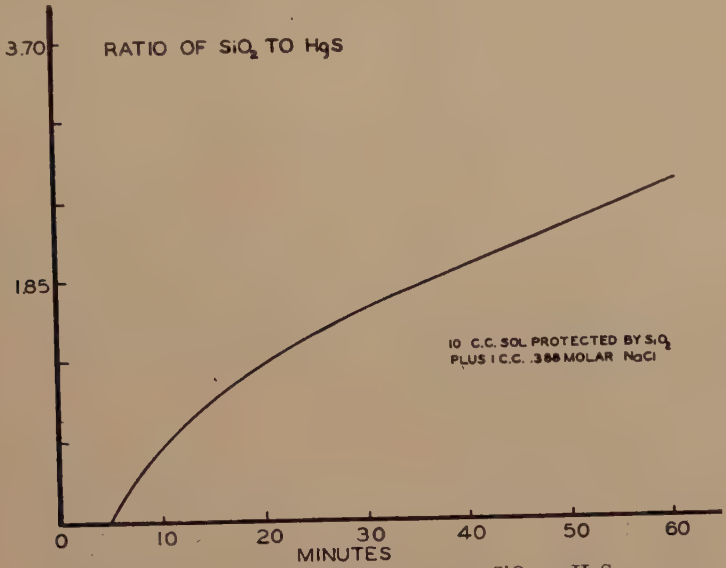

FIG. 3.—EFFECT OF VARYING RATIO OF $SiO_2$ TO HgS.
Sol concentration, 2 grams per liter; electrolyte concentration constant.

## DISCUSSION OF RESULTS

A discussion of the mechanism of transport in solution must remain speculative unless the physical conditions of temperature and pressure, and the composition of the solution are roughly known. In the type of deposit under discussion, Ross[1] has shown the temperature and pressure both to be low. Laboratory experiments at atmospheric conditions will be correct, at least in degree of magnitude.

As to composition, we may approach this factor from analyses given by Becker[2] of hot-spring waters that were depositing or had recently deposited cinnabar. Becker's analysis of the water at Sulphur Bank will be taken as an example. The water has many constituents and in order to obtain a quantitative idea of the precipitating power of the various electrolytes it will be necessary to reduce them to a common basis. This may be done by computing the amount of monovalent electrolyte equivalent to the bivalent or trivalent ion under examination on the basis of the Schulze-Hardy rule; that is, since, as demonstrated above, the cations in order of valence have precipitating power in the ration of roughly 1:100:1000, the amount of NaCl equivalent in precipitating or flocculating power to 0.1 mol of $AlCl_3$ would be 1000 × 0.1, or 100 mols. This has been done in Table 1 for the water from the Hermann shaft at

TABLE 1.—*Flocculating Power of Water at Hermann Shaft*

| Ion | Mols of Cation per Liter | Factor | Mols per Liter of Monovalent Equivalent |
|---|---|---|---|
| $Ca^{++}$ | 0.000677 | 100 | 0.0677 |
| $Mg^{++}$ | 0.000294 | 100 | 0.0294 |
| $Na^{+}$ | 0.037209 | 1 | 0.0372 |
| $K^{+}$ | 0.000632 | 1 | 0.0006 |
| $NH_4^{+}$ | 0.000069 | 1 | 0.0001 |
| $H^{+}$ | 0.005963 | 100 | 0.5963 |
| Total | | | 0.7313 |

Sulphur Bank. Therefore, the precipitating powers of the water at the Hermann shaft could be reproduced in the laboratory by a solution of 0.7313 molar NaCl. Probably this errs in overestimation, as $H_2CO_3$ has been treated as a strong acid, whereas it is a weak acid. Also, the amount of $H_2CO_3$ present in solution at depth probably is less, because a portion of it doubtless comes from acid surface water.

What does this mean in terms of the stability of mercuric sulphide? In Fig. 1, it may be observed that a sol having 1.8 grams per liter is stable to 0.05 mols per liter NaCl. Disregarding the effect of silica for the moment, in Fig. 2 it will be seen that an unprotected sol diluted to 0.06 gram per liter is stable to 0.388 molar NaCl, and it is apparent that dilution to a lower concentration would rapidly increase the stability to the point of stability to 0.7313 molar NaCl; that is, the dilute state alone could account for the transport of colloidal mercuric sulphide in solutions of precipitating power equivalent to those found depositing mercuric sulphide.

When the protective effect of silica is added to the effect of dilution, it is evident that mercuric sulphide could be transported by solutions with much higher concentrations of electrolytes. At a sufficiently high silica concentration, the problem of transport becomes one of the stability of a silica sol.

We have, then, direct evidence of colloidal deposition in the particle size of cinnabar in opalite, and indirect evidence in the stability of colloidal solutions of mercuric sulphide to solutions of electrolytes that approximate those found in the field to be depositing cinnabar. Cinnabar deposits containing finely divided cinnabar or the earthy variety are probably deposited colloidally.

## ACKNOWLEDGMENTS

A large part of the work that forms the basis of this paper was done in partial fulfillment of the requirements for the M. S. degree at the Massachusetts Institute of Technology, under the direction of Dr. E. Hauser, who aided greatly its completion. Dr. Mortimer Bloom very generously gave much of his time, and Mr. C. P. Ross and Mr. Michael Fleischer criticized the manuscript.

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# Exploration on the Stillwater Chromite Deposits, Stillwater and Sweetgrass Counties, Montana

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(New York Meeting, February 1944)

## Abstract

Trenching, sampling, and core drilling in Stillwater and Sweetgrass Counties, Mont., by the Bureau of Mines have delimited over 5,000,000 tons of chromite ore containing more than 20 per cent chromic oxide. The Benbow and Mouat-Sampson mines have been developed and are fully equipped for mining and milling 500 tons and 2000 tons a day, respectively.

## Introduction

The Montana chromite deposits were the first to be explored under the Strategic Minerals Act of 1939. In June of that year, the Bureau of Mines started exploration by trenching, test pitting, sampling and diamond drilling, and completed the project in March 1943. The U. S. Geological Survey cooperated by mapping the geology and logging the cores.

The project was to determine the extent and grade of the chromite ore and to indicate sufficient reserves to justify development and exploitation of the deposits for the national emergency. Exploration has indicated that the district contains the largest and most important reserve of chromite ore in the United States. The Benbow and Mouat-Sampson mines have been developed and a 500-ton concentrating mill has been constructed on the Benbow property and a 2000-ton mill on the Mouat-Sampson property. Concentrates were produced from the Benbow mill three months after the attack on Pearl Harbor, when the nation's major chromite supplies from foreign sources were cut off.

## History

It has been known for 50 to 60 years that some chromite deposits existed in an east-west belt along the north escarpment of the Beartooth Mountains about 25 miles north of Yellowstone Park, but nothing was known of their size and extent until the war demand of 1917 and 1918 stimulated the search. In those years two properties in the Stillwater region were actively developed, but no shipments were made.

The Benbow property was located in 1918 by a local group. The property comprised seven claims along the strike at the east end of the belt. An adit was driven several hundred feet on the Eclipse claim. The seven claims were sold to the Chromium Products Corporation of Livingston, Mont., and were subsequently patented. In 1929, an eastern group obtained an option, installed some equipment, and erected several log cabins. The "crash" in the market in 1929 caused the operators to drop their lease, and the property remained inactive until 1939, when the Bureau of Mines began its exploration.

Fourteen claims were located on the Boulder River in 1918 and subsequently patented by the U. S. Chrome Corporation.

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Presented by permission of the Director, Bureau of Mines, U. S. Department of the Interior. Manuscript received at the office of the Institute Nov. 29, 1943. Issued as TP 1751 in Mining Technology, September 1944.

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This group of claims is known as the Gish property. Two adits were driven in the chromite vein and a small gravity mill was constructed, but no concentrates had been shipped because the war had ended and the demand for chromite ceased.

Numerous reports and examinations have been made on the general geology of the Stillwater complex, the principal ones being by Westgate in 1918,[1] Bevan in 1921,[2] and Schafer in 1935.[3] Dr. J. W. Peoples and Dr. A. L. Howland were in a group from Princeton University that began field work in the area in 1930. Several seasons were spent in studying the Stillwater complex, and a Geological Survey bulletin was published in 1940 on its eastern portion.[4] An examination and preliminary report on the chromite deposits in Stillwater and Sweetgrass Counties were made by a Bureau of Mines Engineer in 1939.[5]

## Physical Features and Communications

The Stillwater chromite occurs in an east-west belt along the north escarpment of the Beartooth Mountains about 25 miles north of Yellowstone Park (Fig. 1). This belt, geologically known as the "Stillwater igneous complex," is in Park, Sweetgrass, and Stillwater Counties, Mont. The igneous complex is 1 to 5 miles wide and is exposed for about 30 miles between the Boulder River and a point 5 miles east of the Stillwater River.

The region comprises plateaus of 9,000 to 12,000 ft. altitude, which are cut by transverse, northward-flowing streams to a depth of 3000 to 5000 ft. Chief among these streams, from east to west, are Little Rocky Creek, Stillwater, West Stillwater, and Boulder Rivers.

The plateau is rolling, with numerous monadnocks rising several hundred feet above the general level. The region shows clear evidence of Pleistocene glaciation; many hanging valleys exist, and glacial till is widespread. A few small glaciers remain, and large areas of marsh and occasional lakes are found on the plateau and in the hanging valleys.

The plateau generally is barren of timber above 9500 ft. Below the timber line, in the canyons and on the lower slopes, timber grows abundantly.

The climate is moderate and invigorating during the summer. The length of the summer season varies with the altitude. The winter months are severe, with heavy snowfall. Surface mining and prospecting are virtually impossible during the winter, except at lower altitudes. Even there deep drifts and high winds hamper work considerably.

When exploration was started, only two areas of the belt could be reached by car; one, the Gish area, 30 miles south of Bigtimber, Mont., and the other, the Mouat area, 42 miles southwest of Columbus, Mont. The eastern end at Benbow and Mouat may now be reached from Dean over 9 and 17 miles, respectively, of good gravel roads recently completed by the U. S. Forest Service. Elsewhere, the deposits may be examined only by trails with grades ranging up to 35 per cent. Trees suitable for mine timber, water and electric power are available in the vicinity of the larger Stillwater deposits.

## The Deposits

The Stillwater complex is composed of anorthositic, noritic, and ultramafic (pyroxene-olivine) rocks, of which the ultramafic rocks contain the chromite-bearing zones. It is believed that the chromite is of magmatic origin, which causes the parallel banding of the chromite concentration zones common to this area.

The assumption is that the chromite deposits were formed at the same time as the surrounding rocks and would extend to as great a depth as the ultramafic zone. From this it would seem that the

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[1] References are at the end of the paper.

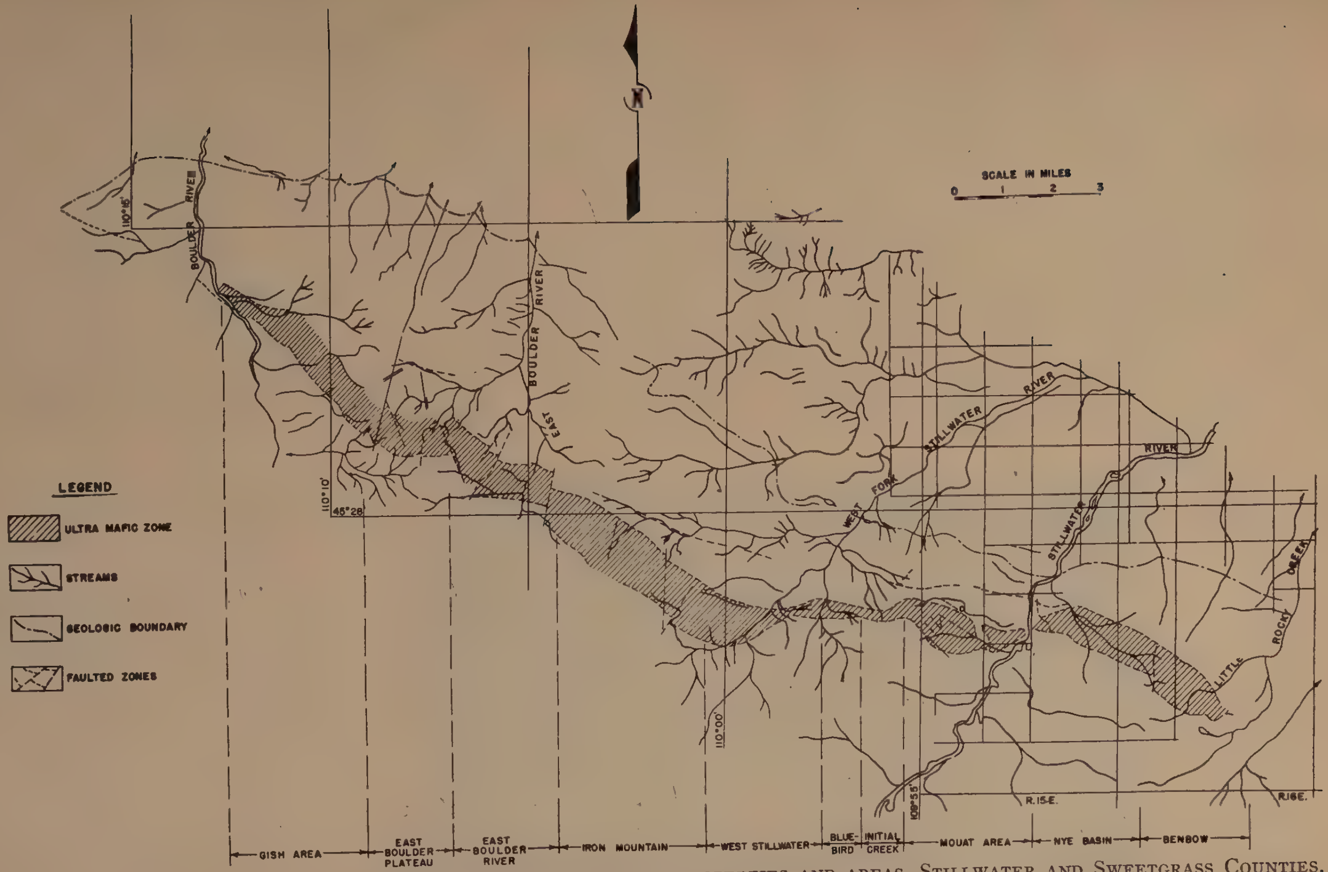

FIG. 1.—STILLWATER IGNEOUS COMPLEX, SHOWING DISTRIBUTION OF PROPERTIES AND AREAS, STILLWATER AND SWEETGRASS COUNTIES, MONTANA.

depth to which mining may go will be limited only by economic factors.

J. W. Peoples and A. L. Howland[4] have described in detail the geology and ore occurrence.

Fig. 2.—Cutting channel sample on cliff above Mountain View Lake, Mouat Area, Montana.

The chromite deposits show various characteristics, but all form steep-dipping layers and lenses parallel to the general layering of the complex. All occur in harzburgite and none has been found in bronzitite.

Chromite has been found for intervals almost 30 miles along the same general horizon in the complex. Zones have been traced continuously for 2000 to 3000 ft., interrupted only by minor cross faulting. However, massive and disseminated layers within the zone seldom continue regularly for more than a few feet or tens of feet. Generally the massive chromite is in a series of overlapping lenses intercalated with lower grade, chromite-mixed harzburgite or serpentine. In places the richest streaks of ore grade into very lean material. The thinner chromite-bearing layers commonly include a rich streak on the footwall with a few inches to a few feet of rock, containing disseminated chromite, above it. There are almost continuous gradations in places from nearly pure chromite to rock with scattered chromite crystals.

The ore mineral commonly referred to as chromite does not conform to the analysis of true chromite, which contains 68 per cent chromic oxide and 32 per cent FeO. Chemical analyses of a number of cleaned mineral composites indicate that the Stillwater chromite contains about 46 per cent chromic oxide, with the chromium-iron ratio ranging from about 1.5 to 2.0. Complete analysis of one representative sample of clean concentrate was as follows: $Cr_2O_3$, 46.09 per cent; FeO, 24.05; $Al_2O_3$, 23.81; MgO, 5.43; $SiO_2$, 1.02; total 100.40.

## EXPLORATION

Exploration was begun on Aug. 21, 1939, and continued until March 27, 1943. Occasional recesses in the work were necessary because of bad weather conditions. During this period, 2304 ft. of trenching, 14,407 ft. of diamond drilling, 75 ft. of shaft sinking and 25 ft. of crosscutting were completed at a total cost of $119,734.05. The ore indicated by this work totals 4,390,000 short tons having an average grade of 21.6 per cent chromic oxide. An additional 1,000,000 tons of ore is inferred from geological considerations. The exploration has not been exhaustive, and probably many millions of tons more will be found, particularly in the eastern part of the belt.

At the time the exploration was started in each locality, it was necessary to set up camps because of the isolation of the areas.

A great deal of preparatory work had to be done, such as building trails, clearing and repairing old trails, clearing land for camps, and building such permanent structures as were necessary. For example, it was necessary to build more than 10 miles of new trails in order to bring in equipment and supplies and pack out samples.

During the early stages of the exploration program, approximately 150 tons of chromite was mined and shipped to Boulder City, Nev., and Salt Lake City, Utah, for experimental work and metallurgical testing. Thus, early information on the metallurgical characteristics of the ore was obtained, which proved useful in devising milling flow sheets.

### *Trenching and Sampling*

Trenching programs were carried out on all the areas, and the chromite was sampled wherever exposed. Trenches generally were excavated across the chromite zones at 50-ft. intervals along the strike and were dug 3½ ft. wide at the bottom and well beyond the walls of the chromite seams. Picks and shovels were used in trenching because the rough, precipitous terrain did not permit the use of mechanical equipment. Each excavation was extended well into the chromite, to make sure it was below any surface disturbance. A few trenches required some drilling and blasting to penetrate beyond the bouldery overburden to solid bedrock. This drilling usually was done by hand-held steel, but a gasoline-operated rock drill was employed with some success.

The continuity of the ore exposures is broken in numerous areas by fault gaps and deep covers of glacial morain or talus slides. In some areas the narrow bands of concentrated chromite are not considered to have any commercial value.

Channel samples were cut across the chromite at the bottom of each trench. In the Benbow and Mouat areas, these channels were 12 in. wide and 6 in. deep, yielding about 100 lb. of chromite per linear foot. Elsewhere on the complex, channels were cut 4 in. wide and 2 in. deep, yielding about 12 lb. of chromite per linear foot (Fig. 2). The maximum width of chromite contained in each sample was about 5 ft. Where apparent changes in grade and character occurred across the band, the length of the sample was reduced to suit the change. In all, 482 samples were taken by channel sampling.

### *Diamond Drilling*

Diamond drilling was begun in June 1940 and continued to the end of the program. At various times, three different contractors were engaged in drilling. Thirty-seven holes totaling 14,407 ft. were drilled, at an average cost of $3.38 a foot.

All the holes except three were drilled from the surface. Two underground holes were drilled from the No. 5 level of the Mouat-Sampson mine and one underground hole from the No. 4 level at the West Benbow mine. In the surface holes the amount of overburden was never predictable and in many places was 60 to 70 ft. deep. Bouldery overburden and broken and faulted areas caused some trouble in drilling, but in general the core recovery was excellent, particularly in ore.

## PACKING AND HAULING

Transportation was difficult and in various places, the equipment and supplies were packed by men, horses, a small tractor with trailer, pneumatic-tired wagons, and trucks.

Pick-up trucks were used to bring in supplies from the supply centers at Billings, Columbus, and Absarokee and to take out samples. Supply points were established at the end of the roads at Dean for the Benbow area and on the Stillwater, West Stillwater, and Boulder Rivers for other areas to the west.

Supplies were hauled 9 miles to camp at Benbow, and the samples were taken out with horse-drawn, pneumatic-tired wagons. The road was steep and passed through glacial moraine, which was bouldery and had not enough gravel to fill in chuck holes. Although comparatively expensive to operate, the pneumatic-tired wagons proved invaluable. Even then, it was necessary to bed the sacks of samples in hay to prevent them from wearing out before they reached the truck depot.

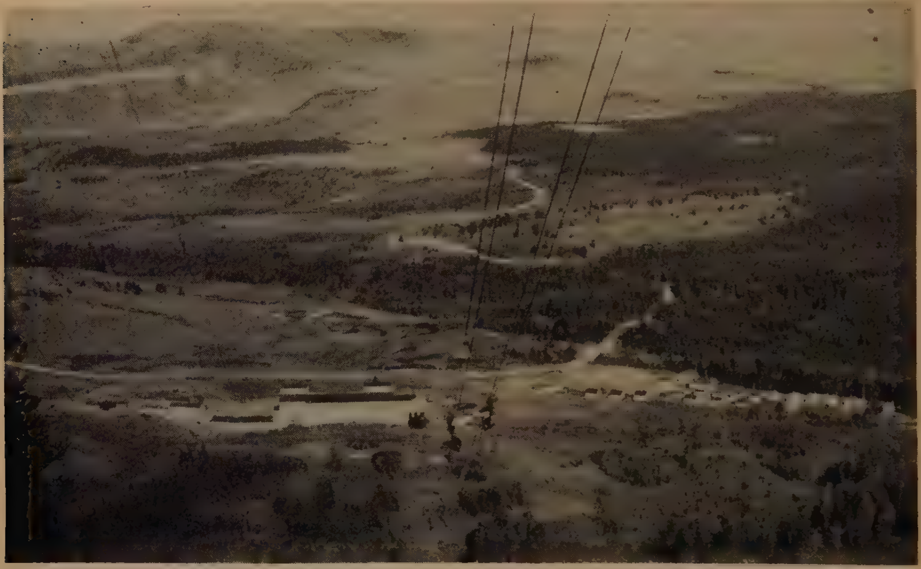

FIG. 3.—BENBOW 500-TON MILL AND AERIAL TRAM.

Pack-horse trains were used extensively for transporting all types of supplies and samples from the road ends to the main camps and return. Supplies packed included such items as picks, shovels, drill steel, diamond-drill parts and rods, sludge barrels, lumber, fire wood, stoves, tents, camp supplies, and samples. Horses were rented at a dollar a day per animal.

At one place, where sufficient work was planned, a narrow, steep, tractor trail was excavated for about 3 miles. The tractor had a narrow gauge and pulled a narrow-gauge trailer having pneumatic tires. The tractor could negotiate the steepest trails and buck fairly heavy snow and mud.

Temporary camps were established whenever the distance to a working site required men to stay out more than one day.

## AREAS INVESTIGATED

### *Benbow Property*

The first deposits on the Stillwater complex explored in 1939 were those of the Benbow area. This area is on the eastern end of the Stillwater complex, at an altitude ranging from 8000 to 9100 ft. During the early stages of exploration, access to the area was 9 miles of poor wagon road up Little Rocky Creek Canyon. Since that time a good, gravel-surfaced road to the Benbow mine has been completed by the U. S. Forest Service.

The exploration program consisted of surface trenching at 50-ft. intervals along the strike over nearly seven claim lengths. This was followed by diamond drilling, to prove the continuity at depth. In June

1941, after the work had indicated a considerable tonnage of chromite, the Anaconda Copper Mining Co., as a Government agent, started exploitation of the deposit. From trenching and diamond drilling, and from the later development, a total reserve of 837,000 short tons of chromite with an average grade of 25.3 per cent chromic oxide was indicated (April 1943). In addition, 258,000 tons of chromite of similar grade is inferred below the present workings.

Since June 1941, the Anaconda Copper Mining Co. has constructed complete camps at the mine and mill for housing the employees. The difficult terrain made it necessary to build the mill several miles below the mine and to construct an aerial tramway for transportating the ore from the mine to the mill (Fig. 3).

The principal feature of the ore deposition in the Benbow area that distinguishes it from others in the district is the extensive faulting in all parts of the Benbow mine. The chromite deposits differ in character along the strike. Near the east end, chromite occurs as layers and irregular lenses 15 to 50 ft. thick in chromite-bearing serpentine. At other places, chromite is concentrated in a single layer 7 in. to 6 ft. thick, with sharp walls, but more commonly layers of relatively massive and disseminated chromite alternate over a width of 2 to 25 ft. Commonly on the hanging wall, the contact is gradational. The footwall contact is generally sharply outlined. The strike is east-west and the dip is steep to vertical. The average minable width is about 5 feet.

Serpentinization of the wall rocks and gangue and extensive faulting have softened and fractured the walls to such a degree that mining is difficult.

### *Nye Basin Area*

The Nye Basin deposits are bounded by the Stillwater River on the west and the Benbow deposit on the east (Fig. 1). They are accessible only by Forest Service trails. The altitude ranges from 6300 ft. at the lip of Nye Basin near the Stillwater to 8100 ft. at the Big Seven fault.

Exploration in the Nye Basin was done in three sections: Big Seven fault, Alice lode and lip of Nye Basin. There is a distance of several thousand feet between each of these sections and the next one, on which no ore is indicated. The basin is a hanging valley and is generally covered by deep overburden.

Trenching and diamond drilling near the Big Seven fault indicated that there are 10,000 tons of ore containing 26.0 per cent chromic oxide. The ore body averages 5.3 ft. in width and is exposed for 190 ft.; the depth is assumed to be 100 feet.

The Alice lode area has an exposure 1000 ft. long, averaging 2.9 ft. in width. It averages 19.6 per cent chromic oxide. The ore reserve, based on the assumption that the deposit continues to a depth of 100 ft., is 29,000 tons.

Exploration near the lip of Nye Basin indicated only a narrow stringer of chromite ore that would have no commercial value.

In consideration of these facts and the inaccessibility of the deposits, the area is not promising, except for the segment near the Big Seven fault, which eventually may be mined from the Benbow workings.

The satisfactory solution of a fault problem in the Big Seven fault section is of particular interest. The West Benbow deposit was traced by trenching to a fault later called the Big Seven. Indications were that the movement was large and to the north, as the trace was indicated by a wide, shallow trough cutting across the slope of the mountain. After considerable investigation, the base of the complex was located on both sides of the fault. By measuring the distance between the base and the chromite and applying this measurement beyond the fault, the chromite was found within 25 ft. of the point indicated, under 20 ft. of overburden. The throw was about 1000 feet.

### *Mouat-Sampson Mine*

The Mouat chromite deposit lies west of the Stillwater River at an altitude ranging from 7900 to nearly 9000 ft. It is about 2000 ft. above the river. This was the second deposit to be explored and has turned out to be the largest known deposit in the United States.

It can be reached by car from Columbus, Mont., the nearest railroad junction, over 30 miles of oiled-surface highway and 19 miles of good gravel road recently completed by the U. S. Forest Service.

Exploration was started in October 1939 and was completed in March 1943. The initial program called for tracing the chromite bands along the surface by trenching and test pitting, and sampling the exposures. With the encouraging

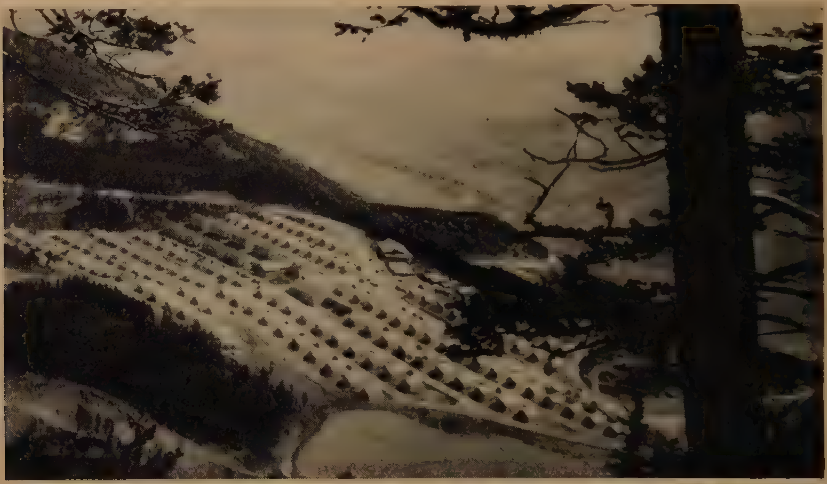

FIG. 4.—CAMP FOR MOUAT-SAMPSON MINE NEAR MOUNTAIN VIEW LAKE.

FIG. 5.—MOUAT MILL NEARING COMPLETION.

results obtained in this initial phase of the exploration, a program of diamond drilling was carried out. Ten holes were drilled for a total of 4687 feet.

In 1941, the Anaconda Copper Mining Co., as a Government agent, started exploitation of this deposit and has since built a 2000-ton mill and townsites at the mine and mill (Figs. 4 and 5).

From the surface trenching and diamond drilling, and from underground development, 3,000,000 short tons of chromite ore having an average grade of 21.6 per cent chromic oxide was indicated (April 1943). In addition, it is estimated that there is 876,000 short tons of inferred ore with an average grade of 21.1 per cent chromic oxide. Sufficient ore has been indicated at the Mouat-Sampson mine to justify plans for several years operation. Future development probably will add several million tons to the present supply.

There are three zones of concentration in the Mouat area; the lower zone contains irregular streaks and pods and some dissemination; the central zone, called the G band, is characterized by alternate layers of high-grade massive chromite and lower grade disseminated chromite, which in places widens to 25 ft. and has an average width of 16 ft. The upper zone, called the H band, averages 5.8 ft., which is a good width for mining. The layers dip 50° to 60° northwest and strike northeast. The G and H bands are comparatively regular and have been exposed almost continuously for 2500 to 3000 ft. along the strike. The present mining operations are confined to these bands, and the nature of the deposit lends itself to comparatively cheap underground methods of mining, since the rocks are unaltered and hard. Faulting is minor, except near the surface.

The first discovery of ore in place on the Mouat G and H bands was made in a cliff above Mountain View Lake. The bands were traced nearly 3000 ft. The limits to the southwest are not yet known. To the northeast, the bands are cut off by the Lake fault. The displacement of this fault is not known, as the surface beyond the fault is covered by tremendous blocky landslides (Fig. 6) for about 1½ miles down to the Stillwater River. The movement along the fault must be more than 1000 ft. However, this problem has not been solved.

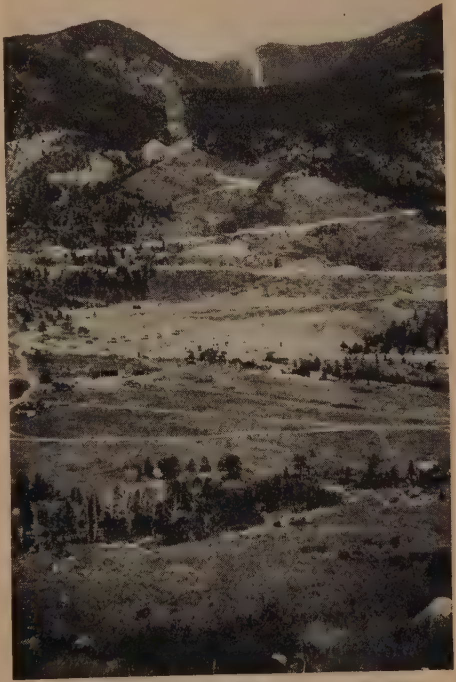

FIG. 6.—MOUAT MILLSITE BEFORE CONSTRUCTION OF MILL SHOWING LANDSLIDES. Clearing for aerial tram and smoke from clearing mine campsite in background.

Sampling along the cliff was difficult. It required the erection of scaffolding, and the use of ropes as protection for the samplers (Fig. 2). Above and below the cliffs, trenching and test pitting were slow, as much of the surface soil contained large boulders.

Diamond drilling was expensive because of the deep, blocky slides, through which drilling had to be done.

### *Initial Mining Claims*

The initial claim area is 2 miles west of the lake camp of the Mouat mine, near the west extension of the Mouat area, at an altitude of 8500 ft. The area was mapped and explored by trenching and sampling. Twenty-four trenches were dug, and 47 samples were taken on the initial claims and adjacent properties.

The average width of the deposit was 6.8 ft., which averaged 13.2 per cent chromic oxide. The layers of chromite were badly faulted, and the segments were short. The total indicated length of the deposit is 490 ft., in three segments respectively 140, 150, and 200 ft. long. The ore reserve, based on the assumption that the deposit continues to a depth of 100 ft., is 33,000 short tons. This tonnage would not justify the construction of a mill, and the faulted condition of the layers would make it impractical to explore them at depth by diamond drilling to increase the tonnage.

The area does not appear to be immediately promising.

### *Bluebird Area*

The Bluebird area is just east of the West Stillwater River, at an altitude of more than 8000 ft., and is inaccessible except over a Forest Service trail.

The trenching and sampling indicate that there is a segment of chromite 600 ft. long, which has an average width of 5.3 ft. and a grade of 19.4 per cent chromic oxide. The remaining 800 ft. of the Bluebird deposit has an average width of 2.5 ft. and an average grade of 17.4 per cent chromic oxide. The latter section presents an extremely difficult mining problem because of its narrow width, and was therefore not considered in computing ore reserves.

Based on the assumption that the deposit continues to a depth of 100 ft., the available ore reserves in the Bluebird area total 32,000 short tons, averaging 19.4 per cent chromic oxide. It is thought that these reserves may be considerably larger than these figures indicate but the inaccessibility of the deposit and its comparatively small size preclude the advisability of further exploration.

### *Taylor-Fry and S. J. Tuttle Claims*

The Taylor-Fry group of claims is one of the oldest locations on the Stillwater complex, but because of the inaccessibility of the area very little exploratory work had been done up to May 1942, when a trenching and sampling program was started. The area extends from the canyon of the West Stillwater River, at an altitude of 6700 ft., to the plateau level and across the plateau at an altitude of 9500 ft. northwesterly for about 4 miles (Fig. 1).

Trenching and sampling indicate that there is a chrome deposit about 3600 ft. long along the canyon side and another 2000 ft. long on the plateau level, with an unexplored gap of about 1000 ft. between the exposures. The pattern of the chromite ore exposed is very regular, showing about 1 ft. of massive chromite and 1½ to 2 ft. of disseminated chromite. The weighted average grade over the exposed length is 16.7 per cent chromic oxide. The ore reserves indicated for the 3600 ft. from the canyon floor to the lip of the plateau for a depth of 100 ft. is 68,000 short tons.

This deposit would be difficult to mine because of its narrowness. There is sufficient ore in sight for a modest mining operation, but not enough to justify construction of a mill.

The S. J. Tuttle claims adjoin the Taylor-Fry claims on the south and were trenched and sampled during the same period. This deposit probably represents a lower horizon of the Stillwater complex than the bands of the Taylor-Fry group and has not proved consistent in other areas where it has been observed.

The trenching and sampling indicate that the chromite deposits on these claims are very erratic in both width and grade and that there is no continuity between exposures.

### *Iron Mountain Area*

In August 1942 a sampling program was started in the Iron Mountain area. This area is in Sweetgrass County, at an elevation of 9500 ft., and is reached over rough mountain trails, which are completely inaccessible during about 9 months of the year, because of extremely cold weather and deep snow.

These deposits were investigated by both the Bureau of Mines and the Geological Survey, and although the chromite exposures seen were small, some sampling was done as a matter of record.

The chromite in the area comprises two narrow bands and a wide disseminated zone. Two trenches on one band showed 2 ft. of 24.6 per cent chromic oxide and 3.4 ft. of 16.5 per cent chromic oxide. The band disappears under overburden and was not found elsewhere. The second band was traced intermittently for a short distance and where sampled showed an average width of about 2.5 ft. with a grade of 18 per cent chromic oxide. The disseminated zone covers an area about 30 ft. wide and contains 8 per cent chromic oxide.

### *East Boulder River Area*

The East Boulder River area, in Sweetgrass County, ranges in altitude from 8400 to 9300 ft., and is accessible only over 12 miles of Forest Service trail. For 9 months of the year, it is inaccessible.

The chromite had been definitely exposed by the owners in three widely separated pits. These pits were channel-sampled, and it is believed that these samples represent the general tenor of the ore in this area. The deposits are narrow and discontinuous, and the tonnage indicated is not large enough to warrant a mining or milling operation. The three pits were sampled on three claims, and seven samples were taken. The average width is 3.3 ft., and the ore contains 22.6 per cent chromic oxide.

### *East Boulder Plateau Area*

The East Boulder plateau area is in Sweetgrass County, at an altitude of 9000 to 10,000 ft. It is inaccessible for 9 months of the year because of deep snow and severe cold weather; during the remainder of the year it is accessible only over steep, precipitous Forest Service trails.

The chromite deposit has been traced intermittently for several thousand feet by outcrops and float. The four best exposures were sampled. The chromite layers were found to be narrow and low in grade.

### *The Gish Property*

The Gish property, near the west end of the Stillwater complex in Sweetgrass County, is considered third in importance on the complex. It is easily accessible from Bigtimber up to the Boulder River. The distance is 33 miles. The deposits outcrop at the river at an altitude of 5500 ft. and have been traced to an altitude of 6200 feet.

Surface trenching was done during September and October of 1941. During this time 61 trenches were dug along the strike at 50-ft. intervals and 186 samples were taken. In addition, two old adits on the property were sampled at 10-ft. intervals.

Early in the spring of 1942, a diamond-drilling program was initiated to explore the deposit at depth. Five diamond-drill holes were put down over a strike length of 2000 ft., covering a vertical range of 500 ft. below the surface.

From the surface trenching and diamond drilling, 403,000 short tons of chromite ore is indicated, with an assay grade of 16.2 per cent chromic oxide. In addition, 119,000 short tons of similar grade is inferred between the bottom of the diamond-drill holes and the flat, west-dipping fault structure found in underground adits.

Exploitation of the Gish chromite deposit was started in the spring of 1942 by the Anaconda Copper Mining Co., acting as a Government agent. Two adits were driven and some mine buildings were constructed.

The chromite in this area is similar to that in other areas on the complex. The layers range from massive bands containing 36 to 40 per cent chromic oxide to disseminated ore containing 10 to 20 per cent chromic oxide. Generally, the zone comprises a massive layer on the footwall and a disseminated chromite on the hanging wall. Analyses of clean concentrates show assays of 43 per cent chromic oxide and a ratio of chromium to iron of 1.8. Surface sampling showed an average width of 5.37 ft. containing 15.1 per cent chromic oxide. The drill holes showed an average width of 4.02 ft. containing 17.8 per cent chromic oxide. The weighted average of the deposit is 4.6 ft., containing 16.2 per cent chromic oxide.

## ACKNOWLEDGMENTS

The writers wish to acknowledge the work done by the various Bureau engineers who at one time or another were responsible for the exploration that made this paper possible; i.e., J. H. East, Jr., J. J. Naughton, E. H. Sharp, R. N. Roby, and R. D. O'Brien. They also wish to thank J. W. Peoples and A. L. Howland, of the U. S. Geological Survey, for their splendid spirit of cooperation during the exploration. The various property owners, local officials of the Anaconda Copper Mining Co., and other Government agencies were very helpful also.

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# Manganese Deposits of Costa Rica, Central America

By Benjamin N. Webber,* Member A.I.M.E.

(New York Meeting, February 1942)

The peninsula of Nicoya contains virtually all of the known manganese deposits of Costa Rica. These are south and west of the Tempisque River, which flows across the peninsula near its juncture with the mainland. This peninsula forms the westernmost part of the province of Guanacaste, which is the northern of the two Pacific Coast provinces of Costa Rica.

The larger villages are connected by ox-cart roads and trails. The area may be covered on horseback without difficulty except during the rainy season, when rivers are in flood and difficult to ford. This is particularly true of the Rio Seco-Guayjiniquil area, where there are no inhabitants and few open trails through the tropical jungle. Santa Cruz and Sardinal are served by the Taca Airlines, two planes a week from San José and Puntarenas. The port of Puntarenas has excellent loading facilities but none are available nearer the manganese districts; there ore must be lightered to vessels standing offshore.

## Physiography

In Costa Rica there are three physiographic divisions: (1) the central plateau (Mesa Central), a somewhat dissected lava platform upon which are superimposed volcanic peaks, some of which are still active; (2) the Atlantic lowlands and (3) the Pacific lowlands. The peninsula of Nicoya and the larger part of the province of Guanacaste lie within the Pacific lowlands. The term "lowland" indicates relation to the central plateau, as the peninsula of Nicoya is largely mountainous. The mountains of Nicoya form the southern extremity of a range of low hills, which are almost continuous along the Costa Rican coast to the Nicaraguan border. The coast line shows evidence of subsidence. The Gulf of Nicoya obviously is the drowned valley of the Tempisque River. The west coast of Nicoya shows many rocks, shoals, bays, inlets and small islands as evidence of a sea encroachment.

The altitude of the manganese deposits ranges from 0 to 800 ft. Topography is of the mature type of wide valleys and low hills, often with abrupt slopes. The climate is tropical and humid; rainfall of 125 in. falls largely between Sept. 1 and Jan. 1. Vegetation often is very dense, the Rio Seco-Guayjiniquil area being truly tropical jungle, containing many hardwoods.

## History and Production

The production of manganese ore in Costa Rica apparently dates from the first World War. It reached a peak of about 10,000 tons in the year 1919, then declined rapidly. Evidence of this period of prospecting has rotted away or been swallowed by the jungle. The war now in progress apparently revived the interest in Costa Rican manganese and has encouraged further prospecting. Deposits long known have been redenounced and trails have been cut through the jungle to them. Many new deposits have been discovered in the last

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Manuscript received at the office of the Institute June 13, 1941. Issued as TP 1445 in Mining Technology, March 1942.

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20 years and some trails have been cut and roads opened but actual exploration has not been extensive.

Manganese-ore production of Costa Rica, taken from reliable sources, is listed in Table 1.

TABLE 1.—*Production of Manganese Ore in Costa Rica*

| Year | Long Tons[a] | Value, Colones[b] |
|---|---|---|
| 1916 | 1,244 | 90,560 |
| 1917 | 7,163 | 442,875 |
| 1918 | 9,680 | 640,069 |
| 1919 | 9,988 | 645,978 |
| 1920 | 3,202 | 209,819 |
| 1921 | | 37,964 |
| 1921–1924 | | None |
| 1938 | 300 | |
| | 31,577 | 2,067,265 |

[a] Imperial Institute Mining Resources Department: Manganese.

[b] V. M. Cabrera: Publicacion de la Secretaria de Gobernacion, Guanacaste, 1924.

## GENERAL GEOLOGY

Some of the manganese deposits of Guanacaste were described by Sears[1] and some helpful information on the sedimentary rocks of part of Guanacaste was recorded by McDonald.[2] The latter gives a stratigraphic sequence of sedimentary rocks that would be next to impossible to work out in the area containing the manganese deposits, because of the predominance of igneous rocks containing isolated roof pendants or infaulted blocks of sediments. The area reconnoitered by the writer is, roughly, that shown on the index map (Fig. 1) and a small part of Nicoya Peninsula near Colonia Carmona.

Over most of this area, the predominating rock is diabase. It is a gray-green, aphanitic rock with few minerals that can be determined with a hand lens. It varies somewhat in composition and texture and is clearly intrusive. Its texture and attitude indicate that it is a shallow intrusion, a laccolith or large sill. The type locality is Rio Seco and a thin section of material from this locality[3] shows laths of basic plagioclase intergrown with augite in an ophytic texture, with smaller quantities of orthorhombic pyroxene and olivine. Magnetite and pyrite often are megascopically visible and magnetite is typical of stream and beach sand throughout areas of diabase.

The Rio Seco diabase is a widespread and probably shallow intrusion. It occurs from the Guayjiniquil River to Culebra Gulf and from Playa Real to Santa Cruz. Contacts of the diabase with intruded sediments were observed at Puerto Viejo, Chapernal and Sardinal. South of Puerto Viejo, the diabase has intruded a siliceous shale with some thin beds of arenaceous limestone. Contact action is feeble and is marked by weak silicification of shale and limestone and by a chilled phase of diabase. At Chapernal, diabase intrudes shale and is heavily pyritized for 10 to 12 ft. from the contact. Near Sardinal diabase intrudes shale and both rocks are silicified near the contact.

A gabbro consisting of plagioclase, pyroxene and ilmenite has a small outcrop near Panama Bay and a similar rock is in fault contact with a highly altered olivine-rich rock, probably peridotite, near Coco Bay. The relationship of these rocks to more prevalent diabase is unknown, as they were not observed in contact.

North of Puerto Viejo some small outliers of extrusive rock rest unconformably on the diabase. Areally they are apparently unimportant. Near Colonia Carmona a volcanic breccia of andesitic aspect occurs in a mass larger than those of the outliers mentioned, suggesting that the locus of origin of these extrusive rocks may not be the same as the extrusives of the central plateau.

Sedimentary rocks are not abundant in the manganiferous area and sediments appear to form isolated roof pendants and infaulted blocks in diabase. Between Puerto

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[1] J. D. Sears: Manganese Ore in Costa Rica. U. S. Geol. Survey *Bull.* 710 (1918).

[2] D. F. McDonald: Informe Final, Geologico de Costa Rica, Tomo Segundo, 1919, Revista de Costa Rica.

[3] Thin Section Report by Arizona Bureau of Mines.

Viejo and the Guayjiniquil River, no sediments were observed. From Puerto Viejo northward they form isolated outcrops. These rocks consist almost entirely of shales; some intercalated, thin, limy beds occur near Puerto Viejo. Most of the shales are very siliceous, grading into thin-bedded chert, which appears to be a sedimentary facies of the shale rather than a replacement product. A few thin beds of fine-grained white sandstone were observed in cherty shale.

McDonald, as a result of studies of the area south and east of the manganiferous terrain shown on the index map, groups the sediments as follows:

Puerto Carballo, arenaceous limestone, Eocene
Unconformity
Nicoya series
  Nicoya limestone
  Colorado limestone
  Sabana Grande siliceous limestone and chert
  Nicoya shale

The shales and cherts observed in connection with the manganese deposits probably coincide with the Sabana Grande and Nicoya formations. The diabase was intruded near the base of the Nicoya series and the upper members were removed by erosion. Doming and consequent fracturing facilitated the removal of most of the sediments above the diabase intrusion.

Faulting is widespread and complicated. Sedimentary roof pendants often are closely folded. At least two periods of faulting are indicated.

A few elements of the geologic history of northwestern Nicoya seem to be indicated. The base upon which the Nicoya series was deposited may consist of basic plutonic rocks (gabbro and peridotite of Ocatal and Panama Bay). The shaly siliceous sediments were deposited in shallow water. The considerable quantity of sedimentary silica (cherts) associated with the shale suggests contemporaneous volcanism. After deposition and induration of the Nicoya series, the Rio Seco diabase was intruded into the Sabana Grande cherts and Nicoya shales as a widespread shallow intrusion, a sill or laccolith. One period of faulting was coincident with adjustments following the intrusion and cooling of the diabase. Erosion has removed most of the Nicoya series, exposing large areas of diabase. Scattered and local volcanic activity took place after at least part of the diabase was exposed. Hypogene manganese minerals form deposits in diabase, chert and shale. Some follow faults and are cut by later faults. Supergene manganese minerals are relatively recent.

## MANGANESE DEPOSITS

Manganese minerals are rather widespread (Fig. 1). Manganese occurrences are of two general types; hydrothermal deposits of hypogene origin, associated with abundant jasperoidal silica and concentrations and superficial replacement deposits of supergene origin.

Of 40 deposits, seven are definitely hypogene, others are doubtful, and the remainder are of supergene concentration type. A deeply decayed and supergene-enriched hypogene deposit so resembles a supergene deposit that classification is difficult, since exploration work is often negligible or lacking.

Hydrothermal hypogene manganese deposits occur in diabase, in cherty shale and on their contact. Hypogene deposits in diabase show a strong similarity of habit. Deposits consist of small flat or gently inclined lenses of manganese oxides and cherty silica. They are commonly crescentic in cross section, the horns of the crescent pointing downward. Often the hanging wall is more acutely arched than the footwall. Two or more lenses may be closely spaced and usually are arranged with their long axis parallel. Manganese oxides easily identified are pyrolusite, manganite, psilomelane; other oxides may be present.

Abundant quartz in the form of dense cryptocrystalline material of cherty or jasperoidal appearance accompanies the manganese oxides. It shows no crustification or banding and commonly is black because of contained manganese oxides. It ranges from manganiferous silica to siliceous manganese ore. Locally veinlets and lenses of white or gray cherty silica occur within a lense of manganiferous silica. A dense cryptocrystalline quartz with sufficient hematite between and replacing quartz grains to produce a red or yellow coloring is also associated with this type of manganese deposit. This jasper, locally called "curiol," occurs in considerable quantity in and adjacent to manganese lenses, where it appears to have formed by replacement of diabase. The quartz grains are anhedral and interlocking and are

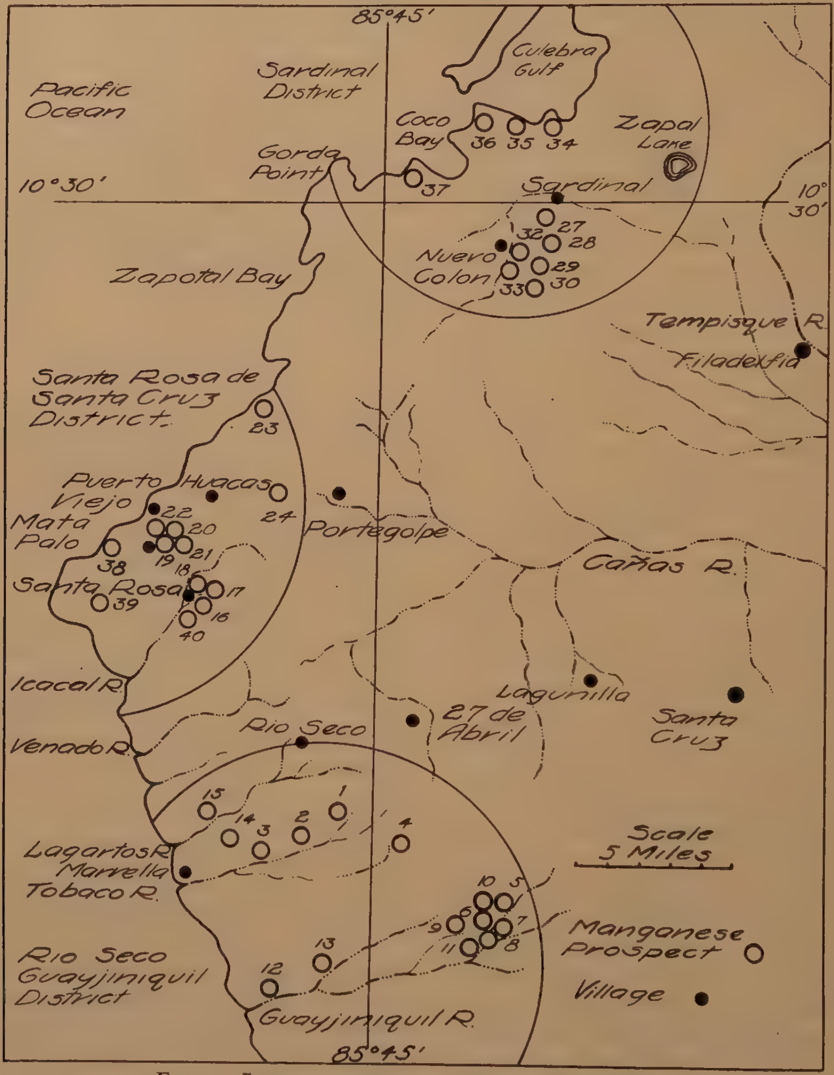

FIG. 1.—INDEX MAP OF PART OF NICOYA PENINSULA.

partly replaced by hematite. The jasper often is cut by later veinlets of white quartz. Cherty material with no iron or manganese coloring shows this phenomenon microscopically. Quartz and hematite are the only observed hypogene gangue minerals. Secular decay of diabase is deep and wall-rock alteration other than silicification is difficult of recognition. Occasionally the jasper is separated from the diabase by a clear line of demarcation but the jasper may grade outward into silicified diabase. Silica predominates over manganese and deposits of silica, both cherty and jasperoidal, conforming in habit to that of the manganese deposits, were observed without any associated manganese. In hypogene deposits entirely within the diabase it is difficult to find any connection between mineralization and faulting. Probably these lenses formed on zones of weakness rather than well-defined faults. The crescentic shape of lenses suggests that doming cracks due to shrinkage in cooling of the diabase may have been followed by the manganese-bearing and silica-bearing solutions.

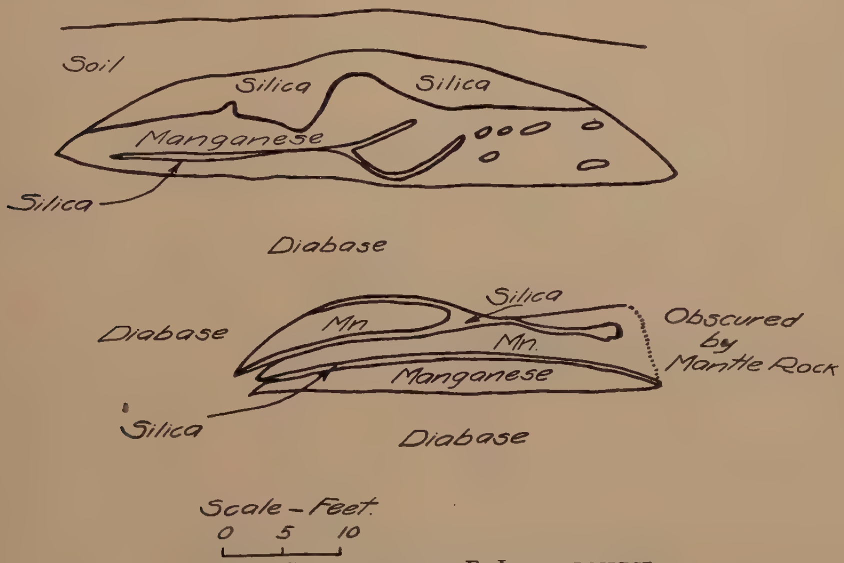

FIG. 2.—CROSS SECTION OF EL LICOR PROSPECT.

Ore from this type of deposit invariably is high in silica, averaging about 25 per cent, with iron 3 to 4 per cent, sulphur and phosphorus negligible. Some ore has been won by sorting surface material that has been enriched by residual concentration of manganese. El Licor deposit (Fig. 2), near Santa Rosa de Santa Cruz, is the best exposed example of this type.

The only deposit with hydrothermal characteristics that occurs entirely within the sediments is the Miramar property. This deposit is one of the few that can be definitely connected with faulting. A series of cherty shale and chert beds striking northeast and dipping 47°NW. are cut by a low-angle fault striking northwest and dipping 35°NE. Considerable jasperoidal silica has been introduced along the fault, mainly on the footwall. Both silica and manganese are restricted to a narrow zone along the fault. Manganese on the fault in the bottom of the old workings is very siliceous and in small quantity. The footwall of the fault forms the slope of a hill where the hanging-wall rocks have been removed by erosion. Undoubtedly, supergene concentration and enrichment have played a large part in the formation of a minable grade of manganese ore from a lower-grade siliceous protore.

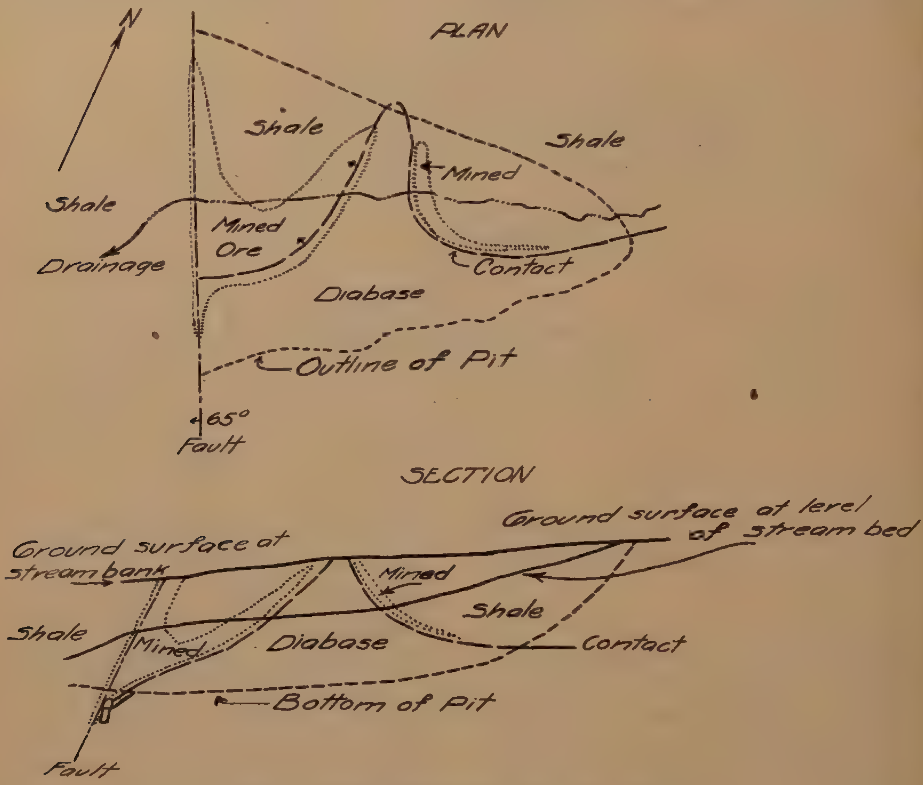

FIG. 3.—PLAN AND SECTION, EL CURIOL MINE.

The Curiol deposit (Fig. 3) apparently is of hypogene hydrothermal origin and occurs partly on a fault contact and partly on an intrusive contact between a rather basic phase of the Rio Seco diabase and siliceous shale. This deposit yielded about 9000 tons and was one of the largest single producers between 1917 and 1920.

An irregular intrusive contact between diabase and shale has been displaced by a fault that brings shale into fault contact with diabase. Silica and manganese were deposited on the fault and on the intrusive diabase-shale contact, but since both diminish on the contact away from the fault, the latter probably was the locus of mineralization. Jasperoidal silica reaches the greatest development in the diabase in the angle between the contact and the fault. Silification and manganese deposition indicate that mineralizing solutions traveling on the fault fissure spread out on the diabase-shale contact. The greatest localization of ore apparently was on their juncture. Enrichment played a considerable role in the origin of the minable ore, as it occurs in a topographic basin and the conformation of the upper surface of the diabase formed an ideal trap for supergene solutions. The fault is mined for a short distance below the contact but ore apparently became more siliceous with depth.

Hypogene deposits have the same associated mineralization, quartz and hematite, as jasperoid, regardless of the enclosing rock. The different environment obviously is a function of structure.

Supergene deposits usually occur in the mantle rock and do not extend into bedrock except through replacement along joint planes or small slip faults. The outcrops usually consist of pebbles and boulders of manganese in the soil or as a crust of manganese nodules over the surface. The horizontal outline of the deposit is very irregular and it is roughly tabular, parallel to the bedrock or ground surface. The manganese tenor decreases rapidly as bedrock is encountered. Bedrock may be either diabase or shale. Manganiferous material from these deposits generally contains more iron and less silica than definitely hydrothermal deposits, although if superimposed on chert or cherty shale it is likely to be high in silica. Pyrolusite and psilomelane are predominant manganese minerals. Where these deposits extend into bedrock on joint planes or slip faults, there is no silicification of wall rock as in hypogene deposits. Silica occurring in this type of ore is fortuitous and consists of admixed sand and pebbles of jasper if the bedrock is diabase, and of chert if the bedrock is shale. Occasionally larger jasper boulders occur with these deposits, indicating formation by secular decay of an area containing hydrothermal deposits.

The source of the manganese of the supergene deposits is not difficult to postulate, as a sufficient number of hypogene manganese deposits are known to occur within the same area, so that it may be safely assumed that secular decay of hypogene manganese deposits furnished the manganese that was transported and emplaced by supergene agencies. It has been established that manganese may migrate as the bicarbonate in solution or as hydrosols of manganese oxides in the presence of organic matter and be precipitated by crenothrix manganifera or similar bacteria, and that precipitation once started by any agency is greatly accelerated by the catalytic action of the manganese oxides already precipitated.

A pronounced deposition of silica, some of which is associated with hematite as jasperoid, in conjunction with the manganese deposition, is taken by the writer as strong evidence of hydrothermal action. The source of the manganese or its method of emplacement is uncertain. Since manganese mineralization is strongly zoned and occupies an outer zone with reference to the locus of intrusion, manganese may be telethermal or epithermal and far from its source. Under these conditions it seems unlikely that the manganese mineralization is related to the diabase intrusion. No traces of manganese minerals other than oxides were observed. It is unlikely that the oxides were the original manganese minerals, although hypogene manganese oxides are known in the mid-Tertiary black calcite veins of the southwestern United States. The original manganese minerals were possibly carbonate or silicates.

# Geology of the Manganiferous Iron-ore Deposits at Boston Hill, New Mexico

By Lawson P. Entwistle,* Member A.I.M.E.

(New York Meeting, February 1944)

## Summary

One of the important reserves of manganiferous iron ore is at Boston Hill, near Silver City, New Mexico. The area consists of a faulted block of gently dipping Lower Paleozoic shale, dolomite, limestone and sandstone, which has been intruded by a roughly circular mass of quartz-bearing monzonite porphyry and subordinate dikes. From a noselike protuberance of the intrusion radiate many premineral faults, which have localized much of the hypogene mineralization.

Three important stages of hypogene mineralization can be recognized: (1) mesitite, a manganiferous iron magnesite, (2) quartz sulphides, and (3) barite-galena. The carbonate stage is widespread but stronger near the intrusion. The quartz may have been derived from hydrothermal alteration of the igneous rocks. Barite-galena is most important in the Chloride Flat mining district, about one mile north, where argentiferous galena provided silver for the supergene ore.

The hypogene carbonate, lean in iron and manganese, has been oxidized and consequently enriched by meteoric waters containing an excess of oxygen and carbon dioxide to form ore bodies of intimately mixed hematite and pyrolusite. The ore bodies, at one place attaining a length of 1600 ft., a width of 300 ft., and a thickness of 25 ft., are largely confined to areas of brecciation and to spreads from faults in Silurian dolomite beneath soft Devonian shale.

## Introduction

Boston Hill comprises three rolling hills southwest of and adjoining the town of Silver City (pop. 5044), Grant County, N. Mex. The mining district covers less than 2 sq. miles. Silver City, on a branch line of the Atchison, Topeka, and Santa Fe Railway, is a supply point for the important mining communities at Santa Rita, Hurley, Hanover, Tyrone, and Mogollon. The Chloride Flat mining district adjoins Boston Hill on the north.

## History and Production

Bonanza-type silver ores were discovered in the Chloride Flat mining district during the silver boom of the seventies and eighties, at which period it has been credited with as much as $5,000,000 production. In 1916 manganiferous iron ore was first shipped from the Boston Hill mining district to Pueblo, Colo., where it was mixed with manganese ore from Leadville and smelted into spiegeleisen. During 1926 and 1927 the Colorado Fuel and Iron Co. mined and churn-drilled part of the district. Mining operations ceased during the depression of the thirties, but production was resumed in 1937. Since 1916 the Boston Hill mining district has produced about 600,000 long tons of ore averaging about 12 per cent manganese and 35 to 40 per cent iron.

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Manuscript received at the office of the Institute Dec. 1, 1943. Issued as TP 1712 in Mining Technology, May 1944.

* American Smelting and Refining Co., Magdalena, New Mexico.

## GEOLOGY

### *Sedimentary Rocks*

A nearly complete local section of the Paleozoic formations is well exposed at the Boston Hill mining district and surrounding area. The sedimentary series, which rests on pre-Cambrian igneous rocks, dips slightly north of east at about 15°.

The Paleozoic section has been beveled by a pre-Cretaceous erosion surface and diminishes in thickness to the west and south. Cretaceous quartzite overlaps formations from Permian in the Hanover-Fierro mining district about 10 miles to the east to pre-Cambrian in the Little Burro Mountains 7 miles to the southwest.

Lower Paleozoic formations are exposed within the mineralized area only. Sedimentary rocks of both Lower and Upper Paleozoic and of Mesozoic age crop out north of the mining area.

*Cambrian System—Bliss Sandstone.*—Paige[1] has correlated a series of red and green sandstones with the Bliss as defined by Richardson[2] in the El Paso, Texas, quadrangle. More recent work[3] has shown that the rock called Bliss sandstone in some parts of Texas is probably of Ordovician age. No new data have been found at Boston Hill, and Paige's original correlation is used.

The full thickness, about 180 ft., is not exposed at Boston Hill, where the outcropping beds are pink to red vitreous quartzite and red and green sandstone. The green mineral has been identified under the microscope as glauconite.[4] Exposures of the Bliss are confined to the west part of the mining district.

*Ordovician System—El Paso Dolomite.*—About 350 ft. of light gray magnesian limestone and dolomite, which has been correlated with the El Paso dolomite by Paige,[1] lies over the Bliss sandstone. The beds at the base are limy and sandy. White to gray chert lenses are common but not abundant.

The outcrops of the El Paso dolomite are largely in the west and southwest part of the mining district. One strip extends southwest from the Boston Hill fault to the Apex fault. The narrow band found northwest of the Boston Hill fault expands rapidly in width to the northwest. Northward the formation thickens to 500 feet.

*Ordovician System—Montoya Dolomite.*—The Montoya dolomite has been divided into three members. The basal beds, which are lenticular and attain a maximum thickness of 90 ft., consist largely of dark gray sandy dolomite. The sandy dolomite beds are overlain by 65 ft. of banded red chert and gray dolomite. Similar beds have been reported existing in many other areas in the southwest part of the state.[5] The uppermost member is essentially a thick-bedded, cherty, gray dolomite, the base of which is a shell limestone with abundant fossils of Richmond age.

South of the Boston Hill fault outcrops of the Montoya dolomite are found throughout the central part of the area. Northwest of the Boston Hill fault the formation has been tilted and squeezed, so that the banded beds alone can be separated from the overlying and underlying formations.

*Silurian System—Fusselman Dolomite.*—The Silurian is represented by about 75 ft. of dark gray dolomite, which is very similar to the upper member of the Montoya dolomite; and only an approximate separation can be made. The weathered surfaces of the beds are characterized in places by irregular pitting, which probably is caused by the differential weathering of the pentameroid brachiopods commonly found in the formation.

The Fusselman dolomite crops out along the east part of the mining district where the formation forms a dip slope.

The dolomite appears homogeneous. A partial analysis of several specimens gives the following average composition: insoluble in HCl, 0.64 per cent; $CaCO_3$, 57.25; $MgCO_3$, 37.12.

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[1] References are at the end of the paper.

*Devonian System—Percha Shale.*—The basal beds of the Percha shale are dark gray to black fissile shale. The upper beds consist of red and green limy shale with nodules of limestone. Along the east slope of the mining district, dispersed on the Fusselman dolomite, are thin scabs of brecciated hornstone cemented by chalcedonic quartz. The entire formation is about 450 ft. thick, of which the upper 70 ft. contains excellently preserved fossils of upper Devonian age. The fossils have been described by Kindle.[6]

Only the basal beds are exposed along the east slope of Boston Hill; the upper beds have been cut out by the Silver City intrusion. Northwest of the Boston Hill fault the formation has been compressed and steeply tilted. The full thickness is found about one mile north in the Chloride Flat mining district.

*Mississippian System—Lake Valley Limestone.*—The basal beds of the Lake Valley are crystalline limestones interbedded with shale. Chert lenses are common. The upper 100 ft. is pure crystalline limestone probably equivalent to the Hanover limestone as defined by Schmitt.[7]

*Pennsylvanian System—Magdalena Formation.*—Only the lower part of the Magdalena formation remains north of the Boston Hill fault, where the beds are bluish gray shales and limestones. The upper part of the section has been removed by the pre-Cretaceous erosion cycle.

*Cretaceous System—Beartooth Quartzite.*—The Beartooth quartzite is composed of about 180 ft. of quartz pebble conglomerate, quartzite, and shale.

*Cretaceous System—Colorado Shale.*—The Colorado shale is poorly exposed. The lower part is gray shale and sandy shale, and the upper part is gray to reddish sandstones and shaly sandstones. Locally the thickness is probably not more than 800 feet.

*Quaternary System.*—To the east of Boston Hill in a small valley is a patch of flat-lying fresh-water limestone with a maximum thickness of about 10 ft. The gravels and sands to the south and west of Boston Hill have been studied by Paige,[1] who believes them to be of Pleistocene age and equivalent to the Gila conglomerate.

## Igneous Rocks

*Pre-Cambrian.*—Pinkish, coarsely crystalline, pre-Cambrian granite crops out along the west part of the mining district. Under the microscope the rock shows a typical allotriomorphic granitic texture.[4] The composition is: albite, $Ab_{95}An_5$, 42 per cent; quartz, 31 per cent; microperthite, 25 per cent; muscovite, 2 per cent. The albite is partly altered to sericite. A few cracks are filled with quartz and specularite.

*Laramide* (?).—The more recent igneous rocks are in the form of sills, dikes, and irregular intrusive masses, which cut rocks as young as Upper Cretaceous. They are intermediate in composition. Elsewhere in the Silver City quadrangle rocks of this type are overlain by mid(?)-Tertiary sedimentary and volcanic rocks.

At Silver City is a roughly circular mass of light colored, porphyritic rock in which phenocrysts of feldspar, quartz, and hornblende are visible. A microscopic analysis[4] shows 41 per cent of the section to be phenocrysts of oligoclase, $Ab_{72}An_{28}$, quartz, and hornblende set in a groundmass of quartz, plagioclase, and an unidentified, greenish ferromagnesian mineral. Hydrothermal alteration has changed most of the plagioclase to sericite; meteoric waters have caused further alteration to kaolinite. The ferromagnesian minerals have been altered to chlorite, carbonate, and magnetite. The quartz phenocrysts are corroded and show rounded embayments. Other thin sections show orthoclase to be a common constituent. The rock is a quartz-bearing monzonite porphyry, and the mass will be referred to as the Silver City intrusion.

Narrow dikes crop out throughout the Boston Hill area, but all have been highly altered by hydrothermal and meteoric waters to sericite, epidote, chlorite, carbonate, and ferric oxide. One sill of diorite porphyry was found in the Chloride Flat mining district and is cut by a hornblende diorite porphyry dike.

## STRUCTURE

The Boston Hill mining district is on the west limb of a syncline of which the east limb is exposed about 10 miles to the east in the Central mining district. No appreciable folding has been found in the Lower Paleozoic rocks at Boston Hill. The area is a homocline. North of the Boston Hill fault, however, the beds have been strongly tilted, and the Colorado shale near the Silver City intrusion has been folded.

Faults dominate the structure. Most of these appear to be premineral and radiate from a noselike protuberance of the Silver City intrusion. The largest is the Boston Hill fault, which, near the intrusion, has a throw of more than 1000 ft. Since the throw decreases rapidly to the southwest, it is hinged in that direction. Some of the loss of throw is due to "horse-tailing," of which one of the branching zones has been called the Fierro fault zone. The second important zone strikes approximately south from the nose of the intrusion. The throws are small on these faults. Many of the fissures have been filled with quartz and minor amounts of sulphides.

Only two large faults, the Apex and the Contraband, do not radiate from the nose of the intrusion. The movement on both of these, at least in part, is recent, for Lower Paleozoic strata are faulted against Quaternary gravels. The Apex fault cuts off the Boston Hill fault and contains as much as 10 ft. of red gouge.

Many faults in the post-ore Tertiary flows, of which the majority strike in a west to northwest direction, have been mapped by Paige.[1] Nearly all the premineral or Laramide (?) faults in the Silver City quadrangle have a north to northeast trend. Some large post-ore faults that strike northeast have been found in the Ground Hog mine[8] in the Central mining district, but these probably represent reopenings along old lines of structural weakness.

## ORE DEPOSITS

The mineralization related to the Silver City intrusion includes both the silver ores at Chloride Flat and the manganiferous iron ores at Boston Hill. The principal ore bodies at Boston Hill lie: (1) near the north-trending faults along the east side of the district and near the Silver City intrusion, and (2) at the intersection of the Boston Hill and Fierro faults and along the Fierro fault zone. One ore body along the east side of the district has been mined, more or less continuously, for 1600 ft. in a northerly direction by a maximum of 300 ft. in an easterly direction. The thickness is about 25 ft. Along the Fierro fault zone some open pits are already over 100 ft. wide, 200 ft. long, and 40 ft. deep; mining operations have not fully limited the size of the ore bodies.

The silver ores mined at Chloride Flat were localized in the Fusselman dolomite beneath the Percha shale by faults and intersecting fissures.

### *Hypogene Mineralization*

The earliest period of mineralization appears to be that of a manganiferous iron magnesite known as mesitite. The mineral has been found only in the deeper workings of the Silver Spot group. It is gray on freshly broken surfaces but soon becomes chocolate brown on exposure. A chemical analysis made by the writer is as follows: Insoluble in HCl, 0.90 per cent; FeO, 16.44; MnO, 3.87; CaO, 1.66; MgO, 29.02.

Miss Jewell J. Glass,[9] upon examining the mineral microscopically, reported the Omega index as 1.789, which corresponds to mesitite.

The carbonate period of mineralization was followed by quartz and sulphides confined to the area near the Silver City intrusion; that is, along the faults with a northerly strike. On the surface and in the shallow workings of the Silver Spot group and adjacent claims, quartz veins and irregular masses of cellular gossan are common. In places leaching has proceeded far enough to remove completely the iron oxides formed by the oxidation of pyrite. Only white honeycombed masses of quartz remain. All the workings with quartz-sulphide veins in place are now under water, but many specimens found on the dump from the workings in the Silver Spot No. 1 shaft were studied in polished section by Dr. Peterson, who reports[4] quartz, pyrite, sphalerite, and galena to have been deposited contemporaneously. Chalcopyrite replaced sphalerite along cleavage cracks. Specimens show narrow veinlets of quartz and sulphides cutting mesitite. Dr. Schmitt called the writer's attention to a small prospect hole in which a sulphide gossan cut manganiferous iron ore.

Barite occurs only sparingly in the Boston Hill mining district; but larger masses and veins are common at Chloride Flat, where the mineral was considered as a guide to supergene silver ores. Barite is associated with argentiferous galena. Carefully cleaned galena assayed 54.96 oz. of silver per ton, but no silver minerals could be identified in polished section. Hypogene quartz is rare at Chloride Flat. There is no direct evidence to show the age relationship between the quartz-sulphide veins at Boston Hill and the barite-galena mineralization at Chloride Flat.

Narrow stringers of pearceite were found in the partings of the Percha shale at the "76" mine at Chloride Flat. The age of this mineral is in doubt. Several veins of manganiferous calcite and quartz occur in the Bliss sandstone along the west part of the Boston Hill area

The common minerals present suggest that the two mining districts are both related to the Silver City intrusion and subordinate dikes. The early mesitite stage, although present in both districts, is strongest near the intrusion. Farther from the intrusion the mineralization is more closely confined to channels. Although larger and higher grade masses of oxidized manganiferous iron ore have been found at the Boston Hill mines than at the Silver Spot mines, the more intense concentration is due to more thorough oxidation rather than to stronger hypogene mineralization. The iron and manganese needed to form mesitite were probably derived from a deep source and replaced the lime molecule in the dolomite.*

During and after the mesitite stage of mineralization hydrothermal alteration was taking place in the igneous rocks, causing a loss of silica and a gain of water and potash. The silica freed by this process was then available for travel and redeposition as quartz veins in the near-by formations. Schmitt,[10] after a careful examination of the Pewabic mine in the Hanover mining district, has been able to show convincingly, supported by many chemical analyses and roughly quantitative mapping of the degree and type of alteration of the igneous and sedimentary rocks, that "the magma need be called upon only to provide the metals, halogens, sulphur, and water"[10] and concludes that the silica is derived from the hydrothermal alteration of the igneous rocks. Similar processes may have taken place at Boston Hill and Chloride Flat as the hypogene quartz with the accompanying sulphides is confined to an area close to the source. The quartz-sulphide stage must be considered weak, as exploration at the principal localizing horizon in the Fusselman dolomite just beneath the Percha shale has failed to disclose commercial bodies of lead, zinc, or copper.

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* Although only mesitite has been identified, it is possible that ankerite was formed in the more limy beds.

The barite-galena may represent a cooler, later period of mineralization more distant from the source. It is also relatively weak, for it followed the faults closely and had only very limited spreads under the Percha shale.

### *Oxidation*

The following oxidized ore minerals have been identified in the two mining districts:

Anglesite
Cerargyrite
Cerussite
Chrysocolla
Embolite
Goethite (?)
Goslarite
Hematite
Jarosite
"Limonite"
Malachite
Plumbojarosite
Pyrolusite*
Smithsonite
Silver (native)

The vertical extent of commercial manganiferous iron oxides is dependent on the completeness of oxidation and the consequent enrichment and concentration of the lean hypogene carbonate due to the removal of valueless material. The effective oxidizing power of acid surface waters in the reactive environment of limestones and dolomites is shallow, therefore the ore bodies are high grade. Most of the known ore bodies are within 75 ft. of the present surface.

The field evidence suggests that only small quantities of sulphide minerals were available for oxidation and the production of sulphuric acid. Experimental data on oxidation of mixed iron and manganese compounds[11] indicate a lag in the precipitation of manganese when the two elements travel in sulphate solution. Only minor segregations of hematite and pyrolusite were noted in the field; usually the oxidized ore is an intimate mixture of the two minerals. Some "limonite" occurs in the gossans from the quartz-sulphide veins.

Experiments by Savage[12] indicate that carbonated waters are a more effective solvent of manganese carbonate and manganous oxide than oxygenated waters. The manganese is transported as a bicarbonate and precipitates as soon as the solution becomes slightly alkaline. Further experiments[11] show that, when oxygen and carbon dioxide are together in considerable excess in water, ferric oxide and manganese dioxide precipitate together. In the presence of the reactive host rock and hypogene carbonate the excess carbonic acid reacts with calcium and magnesium carbonates to form soluble bicarbonates. The solution becomes alkaline and coprecipitation is effected with little travel. The speed of action is shown by the discoloration of the hypogene carbonate after a few years of exposure on the mine dumps. When oxygen and carbon dioxide are in small quantities in water, ferric oxide forms first and manganese later; and resultant oxidized minerals are again segregated.

Experimental data and field evidence suggest that: (1) sulphuric acid waters played only a minor part, as the quantity of sulphides available for oxidation is small and there is little segregation of the iron and manganese oxides; (2) haloids found at Chloride Flat may have derived the chlorine and bromine from land-locked basins[13] or from the atmosphere;[11] and, (3) the intimate mixture of iron and manganese oxide is probably caused by the action of an excess of oxygen and carbon dioxide in

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* Pyrolusite has been confirmed by X-ray and optical studies by Michael Fleischer and W. E. Richmond: The Manganese Oxide Minerals—a Preliminary Report. *Econ. Geol.* (1943) 38, 285.

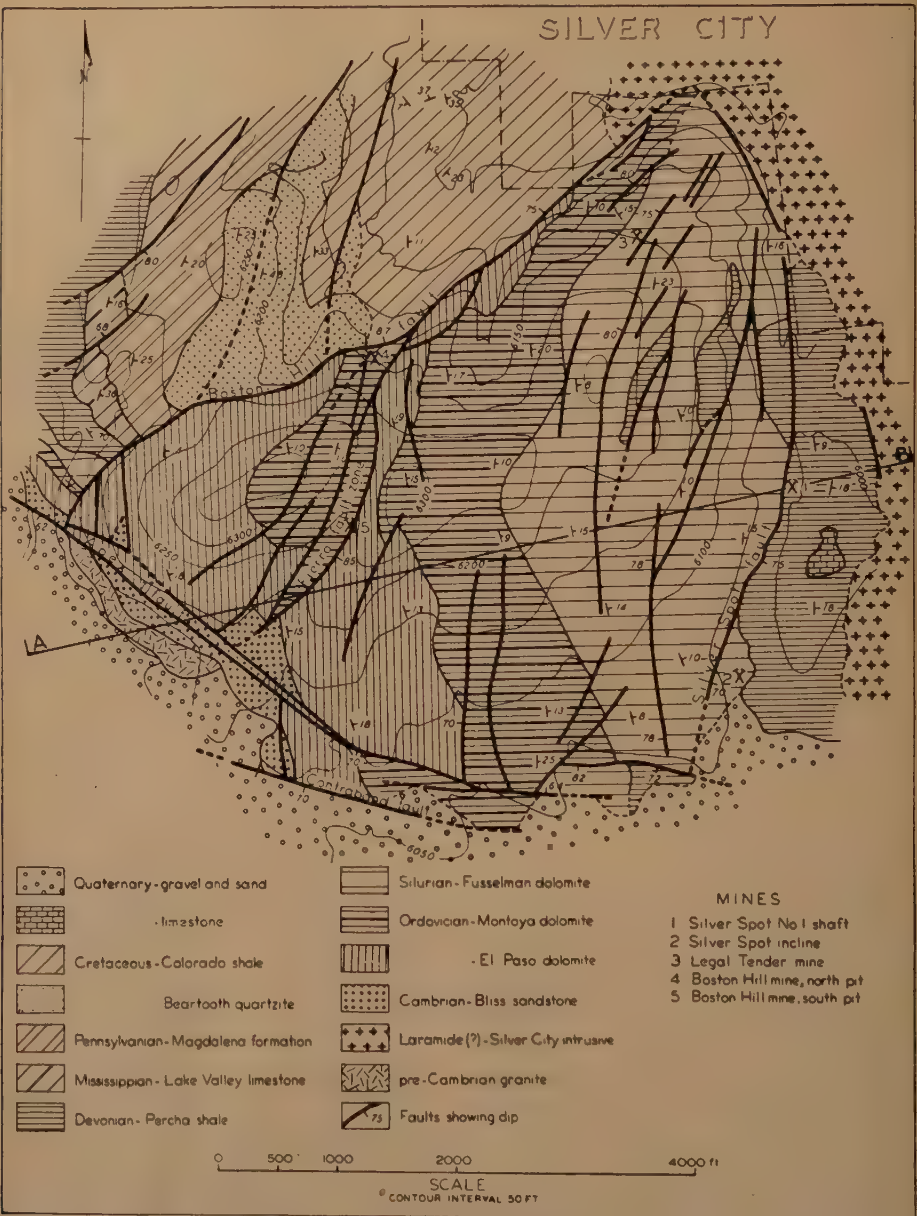

FIG. 1.—AREAL GEOLOGY OF BOSTON HILL MINING DISTRICT, GRANT COUNTY, NEW MEXICO.

meteoric waters on the hypogene carbonate. Transportation as a bicarbonate was minor as alkaline compounds necessary for precipitation were present. Analyses of ore and mesitite show a concentration ratio of about 1:3.

The oxidation of mesitite led to a loss of volume and post-ore subsidence. The collapse gave rise to bedding-plane cracks, which have been filled with post-ore, post-oxidation calcite. Some of the calcite is tinted pink from small included particles of hematite.

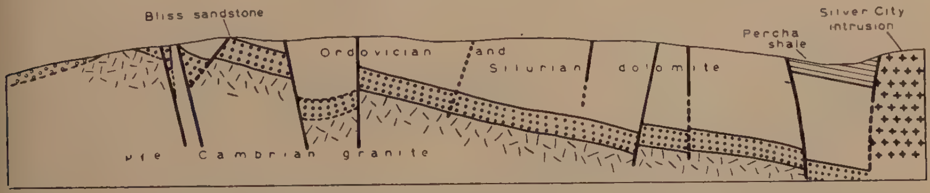

FIG. 2.—SECTION ALONG LINE *AB* OF FIG. 1, LOOKING NORTH.

At the south end of the Silver Spot group the surface is covered with boulders of monzonite porphyry and sedimentary rocks which have been cemented by a caliche-like material. The capping at the mines along the Fierro fault zone is a dark brown soil.

### *Localization of the Ore Bodies*

Thorough ground preparation in the form of brecciation appears to have localized all the ore bodies. Differential ground preparation has caused spreading in certain beds. Ore bodies in highly brecciated ground require less favorable wall rock; this is well illustrated at the Boston Hill mines (Fig. 1, mines 4 and 5). One ore body has formed at the intersection of the Boston Hill and Fierro faults, another in the strongly broken area along the Fierro fault zone. The two zones, with their many subsidiary faults, have produced an elongated area of brecciation. Even the banded beds of the Montoya dolomite are replaced by manganese and iron oxides. In this area, however, there have been opportunities for considerable concentration, for at the time of mesitite deposition there was several hundred feet of Lower Paleozoic dolomite above the present surface. Erosion, concentration, and thorough ground preparation have produced large bodies of good-grade ore. Reports indicate that the ore from this area runs about 12 to 13 per cent manganese and 40 per cent iron.

The second type of deposit predominates at the Silver Spot group (Fig. 1, mines 1 and 2). The ore occurs in the Fusselman dolomite near the overlying Percha shale. Differential ground preparation results from faults in brittle rocks, such as the Lower Paleozoic dolomites, encountering softer overlying rocks, such as the Percha shale. The movement on minor faults in the dolomite is absorbed by the shale in the form of rolls. This feature is well shown in many workings (Fig. 3). Only the larger faults penetrate the shale and reach the overlying limestones. In the areas of intense mineralization and recurrent intramineral faulting this condition probably would not be so effective a trap as at Boston Hill. The ore bodies in this area are lower grade because: (1) the hypogene carbonate was protected for a longer period of time by the overlying shale, and (2) there was little or no favorable ground above from which additional manganese and iron could be removed and reconcentrated.

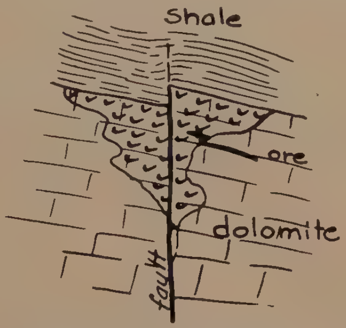

FIG. 3.—SPREADING AND LOCALIZATION OF ORE BENEATH SHALE.

### *Ore Reserves*

A churn-drilling program was carried out on the Silver Spot group in 1926 and 1927. It has been reported[14] that the work developed 500,000 long tons of ore averaging 9 to 11 per cent manganese, 33 to 35 per cent iron, 0.015 per cent phosphorus, 7 per cent silica, and 2 to 3 per cent alumina. An additional 1,000,000 long tons was shown as probable ore. It was noted, however, that mining operations in 1929 and 1930 on the property produced about 82,000 long tons averaging about 8.5 per cent manganese and 27 per cent iron.[15]

There are geologic prospects for as much as 5,000,000 long tons of about 10 per cent manganese and 30 to 35 per cent iron in the district. The Boston Hill group from 1937 to 1940[16] produced about 91,000 long tons that averaged nearly 13 per cent manganese and over 38 per cent iron.

### PRESENT OPERATIONS

The Silver Spot group was operated last in 1930. All the mining was by underground methods. At most places the back is strong enough to allow extraction by room-and-pillar methods.

During the past seven years all the production has come from the Boston Hill group, which is being operated by the Luck Mining and Construction Co. Mining operations have been carried on in open pits (Fig. 4). The ore is hard and must be drilled and blasted. The pits, which in places are more than 50 ft. deep, are arranged so that ore can be picked up by power shovel and loaded directly into trucks. The ore is hauled about 2 miles to the ramps at the Santa Fe Railway at Silver City and shipped to Pueblo, Colo. During the spring of 1943 about 7,000 tons a month was mined

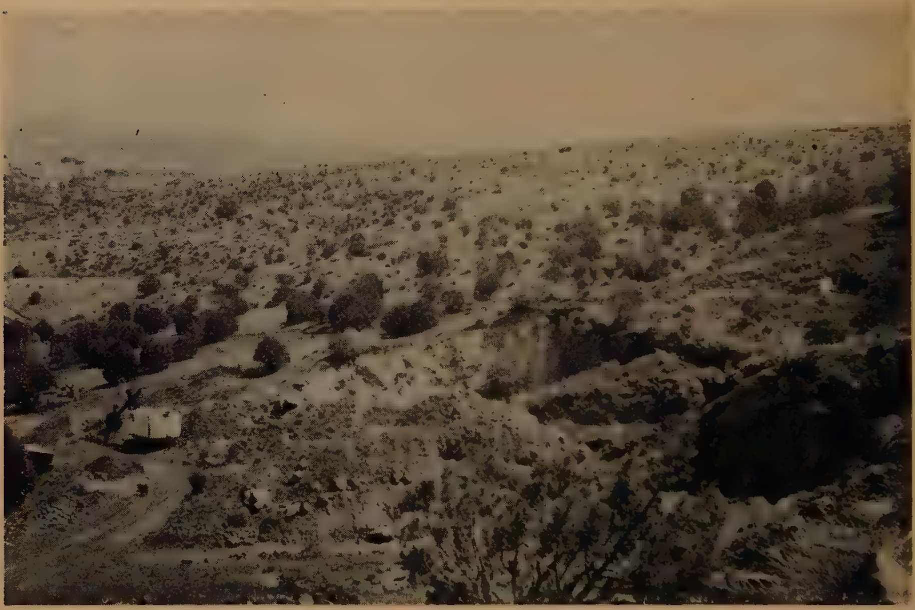

FIG. 4.—MINING OPERATIONS, BOSTON HILL MINES, NORTH PITS.

## ACKNOWLEDGMENTS

The writer thanks Dr. N. P. Peterson, formerly geologist for the New Mexico Bureau of Mines and Mineral Resources, for making microscopic examinations of thin sections and polished sections; G. S. Entwistle, D. R. Purvis, and F. S. Cook for assistance in the field; Dr. Harrison Schmitt for reading the manuscript critically and for advice based on his observations in the area; and Dr. B. S. Butler for his critical reading of the manuscript.

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## DISCUSSION

*(C. D. Hulin presiding)*

C. H. BEHRE, JR.*—In the course of the discussion of this excellent paper at this meeting the general question has been raised as to why manganese oxides are concentrated in certain parts of the oxide zone in such a way that the oxidized manganese and iron ores are separated, despite the fact that the two metallic elements are otherwise so similar in their chemical behavior. I am somewhat acquainted with the occurrence of the oxidized ores of manganese and iron in two mining districts where conditions are strikingly dissimilar and would like to discuss these occurrences and their bearing on this problem.

At Leadville, Colo., the dominant primary manganese source mineral is manganosiderite. The relations between the oxidized iron and manganese minerals at Leadville have not been closely studied but their general position, especially of the high-grade manganese ores, suggested to Loughlin[17] that the manganese travelled farther in the ground water than did the iron. It is clear, then, that downward transportation in solution separated the two metals, but why there should be this separation is not evident. At first glance it might be assumed to be related to the presence of sulphate ions, sulphides being so prominent a constituent in the ore.

However, much the same relations are observed in the oxidized ores of iron and manganese in the Southern Appalachians, where sulphides are rare or absent and therefore can not have yielded the sulphate ions which might have caused the separation. Thus, at Bumpass Cove, Tenn., P. B. King, John Rodgers, and their associates have made a study of the occurrence of iron and manganese.[18] Again the manganese oxide ores occur in general below the iron oxide deposits. Yet here whatever sulphides are present are areally some distance away from the noteworthy occurrences of manganese and iron oxide ores.

The contrast between the behavior of manganese and iron oxides would seem best attributable to two facts: (1) the greater solubility of manganous bicarbonate over ferrous bicarbonate, and (2) the higher percentages of iron in the double carbonate from which these two otherwise largely dissimilar manganese deposits were derived. The percentages of iron in the source mineral—manganosiderite—at Leadville are generally higher, as indicated by the Argall analyses quoted by Loughlin (*Op. cit.*, p. 153), but not uniformly so—not enough so as to justify unqualifiedly the conclusion that *initial ratios* of iron to manganese in the unweathered rock would cause earlier supersaturation of the descending solutions with iron and thus the deeper transport of the manganese. On the other hand, it is difficult in the present state of our knowledge to draw dependable conclusions about the *contrasting solubilities* of ferrous and manganous bicarbonates in the depositing solutions, because the other constituents are not known. Yet the fact that manganese oxides generally occur at greater depths seems to be most readily attributable to such contrasting solubility. The experimental work reported by Zapffe[19] suggests that delayed hydrolysis of the soluble manganese salts in comparison with soluble iron salts may well be a contributing factor; it may indeed be the underlying cause for this contrast in solubilities.

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* Columbia University, New York, N. Y.

[17] U. S. Geol. Survey, *Prof. Paper* 148, 221–222.

[18] Tennessee Geol. Survey *Bull.* 52, 50, 57, 263–264.

It is to be hoped that the recent careful study by W. S. Savage[20] will shortly be supplemented by comparative studies of manganese and iron solubilities in carbonate solutions.

C. D. Hulin*—Recent studies of diamond drill cores from a manganese deposit in Maine yielded unquestionable evidence of secondary enrichment caused by downward migration of manganese.

The deposit was sedimentary in character, the host rock consisting of thinly laminated shale interbedded with thin layers of chert. In the deeper cores (below 40 ft) the principal primary manganese minerals present were neotocite and psilomelane. Sulphides were strictly absent and carbonates were present only in minute quantity.

From the surface to a depth of 40 ft (presumably the position of the water table) effects of oxidation were strongly represented. The neotocite showed varying degrees of advanced alteration, in place, to a very fine grained oxide, probably manganite. Frequently the oxidation of the neotocite was complete. The psilomelane, however, remained unaffected. The oxidized ore was further characterized by innumerable minute crosscutting veinlets of pyrolusite whose distribution was such as to indicate that the dioxide had been deposited from solution along fractures cutting the rock body, rather than by oxidation of adjacent manganese minerals.

Assays of the oxidized ore showed a manganese content approximately 50 pct greater than that of the deeper unoxidized ore. It seems clear that this enrichment of the oxidized ore could only have resulted from downward migration of manganese from portions of the deposit now eroded away, and that the manganese so introduced was precipitated more or less entirely as pyrolusite. In the absence of sulphides, the enrichment apparently involved only normal surface waters of oxidizing character.

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[19] *Econ. Geol.* (1931) **26**, 799–832.

[20] *Econ. Geol.* (1936) **31**, 279–297.

* University of California, Berkeley, Calif.

# Replacement Hematite Deposits, Steep Rock Lake, Ontario

By Hugh M. Roberts,* Member, and M. W. Bartley,† Junior Member A.I.M.E.

(New York Meeting, February 1942)

## Abstract

SUBSTANTIAL deposits of Bessemer hematite have been found recently by drilling beneath Steep Rock Lake, Ontario, which is situated in the northern part of the Lake Superior Region. It will be practicable to divert the waters flowing into this lake and develop open pits on the upper part of two large ore bodies by damming off and pumping out the water in the part of the lake in which they occur. Ultimately, the bulk of the ore will be mined by underground methods. Owing to the generally massive nature of the rocks in the area, no underground flows of major consequence are anticipated after diversion of the lake.

The Steep Rock deposits are attributed in origin to hydrothermal replacement. They were formed, for the most part, along an unconformable, brecciated contact between the Steep Rock limestone and an overlying younger volcanic series. The replacing solutions came in as igneous after-effects, following upon the intrusion of basic igneous rocks that invaded the hanging-wall volcanics. It is believed that this took place in a geological period roughly equivalent to late Middle Huronian, which was the period when the most extensive iron-bearing rocks of the Lake Superior Region were formed.

Comparison is made with other deep-seated iron-ore bodies in the Lake Superior Region, the origins of which are ascribed in the discussion to hydrothermal replacement.

## Introduction

Under the waters of Steep Rock Lake, Ontario, situated on the Canadian National Railway, 142 miles west of Port Arthur, the presence of three ore zones containing red and brown Bessemer hematite has been established. These were discovered and have been partly explored by means of diamond and churn drilling. The positions of the ore zones, designated as *A*, *B* and *C*, are shown on the accompanying maps and cross sections. This exploratory work, begun in 1938 by Joseph Errington* and Julian G. Cross, has been carried on by Steep Rock Iron Mines Ltd., and was continued during the winter of 1941–1942.

For more than 60 years explorers have known that large quantities of hard hematite float of good grade existed along the south shore of Steep Rock Lake, and the presence of ore beneath the lake was inferred by many early geologists and explorers. Despite numerous previous attempts to find ore in place, actual discovery awaited the recent period of intensive exploratory effort in Canada.

The results obtained in the drill holes put down since 1938 by Steep Rock Iron Mines Ltd. are not only of interest, in themselves, to those concerned with the development of ore deposits, but, taken in connection with the surface exposures of adjoining formations, they form the assemblage of facts from which the geological reasoning in this paper proceeds; accordingly, they are set out here as briefly and exactly as may be.

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Manuscript received at the office of the Institute Nov. 24, 1941; revised August 1942. Issued as TP 1543 in MINING TECHNOLOGY, Jan. 1943.

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* The authors wish to acknowledge their indebtedness to Joseph Errington, President of Steep Rock Iron Mines Ltd., and to the Directors of that company, for their assent in February 1942 to the publication of this paper. Mr. Errington died on Feb. 8, 1942.

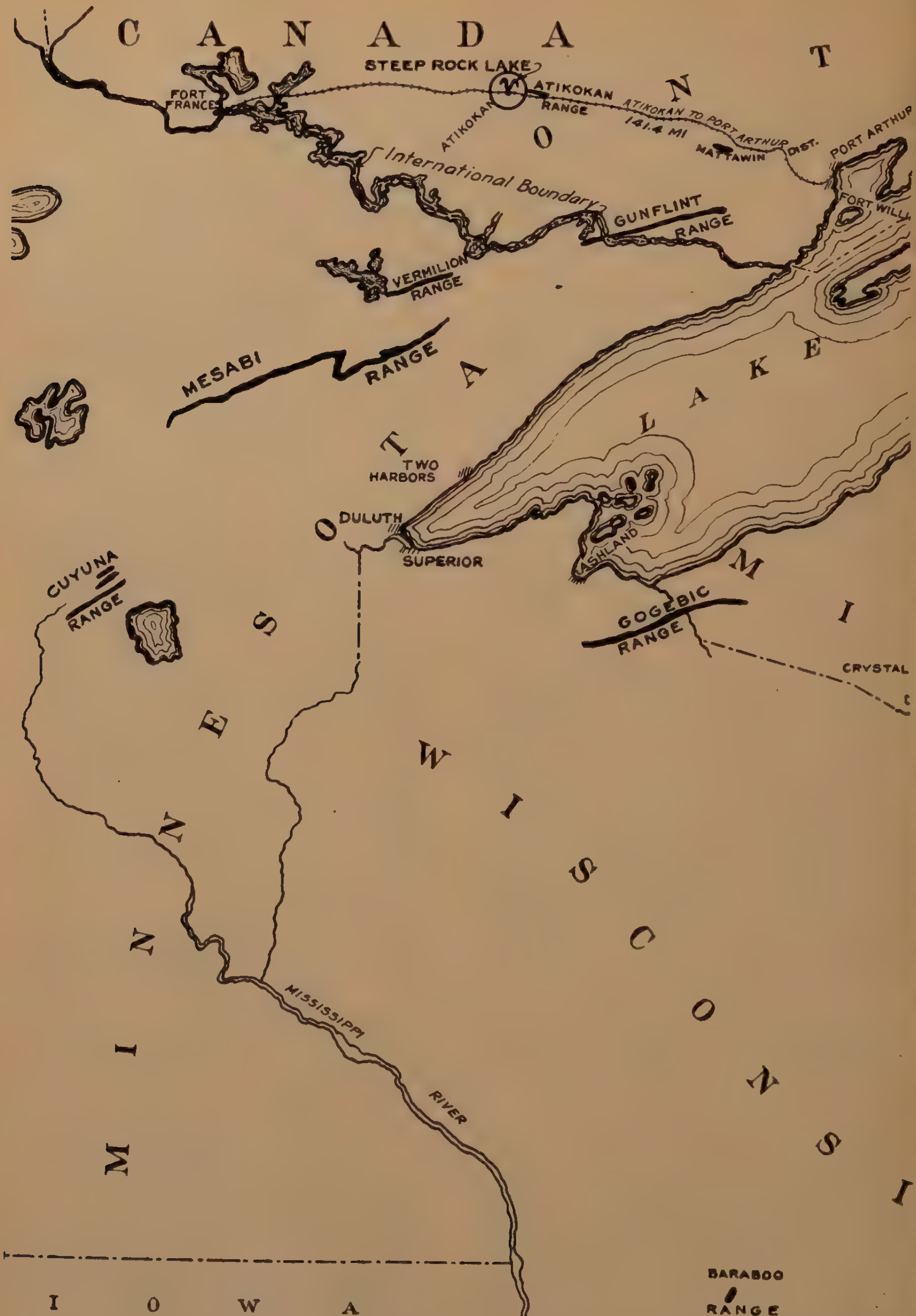
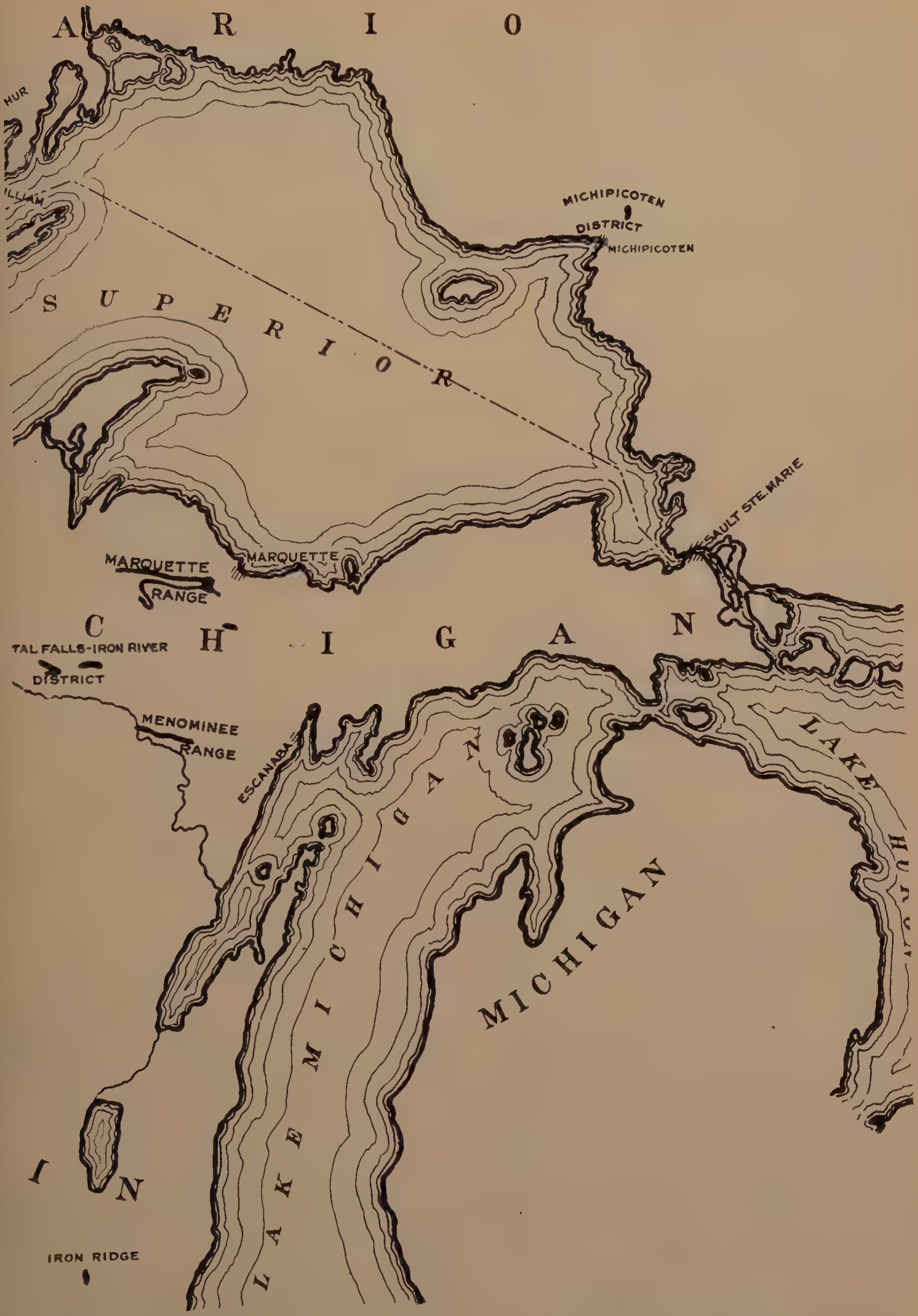


Fig. 1.—Locations of iron
Scale is approximately



RANGES, LAKE SUPERIOR REGION.
1⅛ inches equals 60 miles.

### "Scout" Diamond-drill Holes

Early in the course of the undertaking, it appeared that some plan of broad reconnaissance would be necessary to permit a rapid view of the geology in horizontal section and to obtain a view of the actual extent of the ore zones before deeper work could be projected to advantage. The difficulties encountered in diamond drilling, and the urgency of getting results before the ice would move, also commended this course. Accordingly, Joseph Errington, with a true explorer's instinct, declined to work in detail at first, but, spreading out the drills as rapidly and advantageously as might be, put down some 250 vertical "scout" diamond-drill holes. These holes started from the ice, passed through water and glacial material at the bottom of the lake and penetrated ledge to an average depth of 15 ft. Of necessity, they were not cased; the core recovery averaged only 10 per cent and sludge could not be effectively recovered. Notwithstanding these difficulties, an unexpected and singular uniformity of iron content appeared in the core samples over large areal extents. In 43 holes thus drilled on the *A* and *B* ore zones, these core samples averaged 61.12 per cent iron, 0.031 per cent phosphorus, 3.71 per cent silica and 0.038 per cent sulphur (dried ore, arithmetic average). The *C* ore zone was very scantily explored; only six scout holes penetrated ore at bedrock, but the ore so disclosed is apparently of the same average grade.

During the progress of this preliminary scout drilling, geophysical surveys, made under the direction of A. A. Brant, of the University of Toronto, were most useful in determining geological contacts, and this work gave direction to the choice of positions for the scout drill holes.

### Angle Diamond-drill Holes

A few angle diamond-drill holes were then put down, with varying success; they were drilled from shore and from underground on the 800-ft. level of an exploratory shaft. In one of these holes, No. 27, on the *A* ore zone (Figs. 2 and 3), which went to a depth of 1450 ft. vertically below the lake level, 78 ft. of Bessemer hematite were intersected at an average vertical depth of 1375 ft. This ore contained 62.68 per cent iron, 0.017 phosphorus, 2.90 silica and 0.011 sulphur (combined analyses of core and sludge). While the dip of the ore is not definitely known at this point, the probable true width is 70 ft. This hole was carefully drilled and conditions permitted casing at 5-ft. intervals, thus a correct sample of the ore was taken at that place.

Angle hole No. 280, drilled from underground into what appears to be a hanging-wall lens of the *A* ore zone (Fig. 3)* was equally well drilled. Owing to a mistaken theory as to the nature of the ore, this hole was not completed, but was stopped in ore. Within a 106-ft. section measured along this drill hole, 76 ft. of hematite were cut that averaged 62.60 per cent iron, 0.017 phosphorus, 2.96 silica, and 0.030 sulphur (dried ore). In this, there are two sections of waste—one of unreplaced greenstone and the other hematite containing some visible pyrite—that were not calculated into the averages given above. Hole No. 281, drilled into the *B* ore zone (Fig. 4), was successfully and adequately completed. Ore of the same grade was intersected. But for the most part, diamond drilling, particularly in angle holes, was so impeded by difficult ground that the securing of representative samples was not possible.

### Mesabi-type Churn-drill Holes and Diamond-drill Holes

The general location and approximate grade of the ore zones having been disclosed by the scout drilling and the angle holes, plans were made to prove grade and tonnage of the deposits and to establish their actual configurations. It was believed that this could be done best by means of churn drilling and diamond drilling combined, as worked out and accepted for the Mesabi Range, Minnesota. Since Feb. 1, 1941, thirty-eight such vertical holes have been drilled into the *B* ore zone, 35 of which were in ore. A plan and typical cross sections of these are shown in Figs. 4 and 5. Four holes of this type were put down on the *A* ore zone (Fig. 2). Dependent upon the depths to which they were drilled and their positions with respect to irregularities in the walls of the ore zones, these churn-drill holes cut varying intersections of ore up to 410 ft. on the *B* zone (Fig. 5), and intersections of ore up to 220 ft. on the *A* zone.

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* Drawn by M. S. Fotheringham, General Superintendent of Steep Rock Iron Mines Ltd., to whom the authors acknowledge their obligation, not only for this, but for much material aid in shaping many of the ideas expressed in this paper.

### SUMMARY OF ANALYSES

Summarizing the analyses of samples obtained by the various types of drilling utilized, and comparing them with the analysis of float ore found on Float-ore Island, south of the *B* ore zone, shows an unexpected uniformity of grade (Table 1).

TABLE 1.—*Average Analyses, Dried Ore*

| Type of Drilling | Analysis, Per Cent | | | |
|---|---|---|---|---|
| | Fe | P | $SiO_2$ | S |
| Scout drilling, 10 per cent core recovered. Average arithmetic analysis[a] | 61.12 | 0.031 | 3.71 | 0.039 |
| Ore sections in effective angle holes,[b] core and sludge combined. Weighted average by 5-ft. sections (344 ft. in 4 holes)[c] | 61.46 | 0.016 | 3.00 | 0.016 |
| Ore sections in Mesabi-type churn-drill holes (weighted average of cuttings by 5-ft. sections. 3662 ft. in 38 holes)[d] | 61.14 | 0.022 | 2.96 | 0.038 |
| Float ore from Float-ore Island[e] | 61.42 | 0.031 | 2.79 | 0.060 |

[a] Analyses by J. W. N. Bell, J. T. Donald, Milton Hersey and Lerch Bros.

[b] See detailed sections for particulars respecting angle holes and churn-drill holes.

[c] Analyses by Lerch Bros.

[d] Analyses by Lerch Bros.

[e] Analyses by Bureau of Mines, Ottawa.

Composite samples, taken in proper proportions from all the churn drilling in ore to date show the Bessemer grades listed in Table 2.

TABLE 2.—*Bessemer Grades*
PER CENT

| Constituents | *B* Ore Body (673 Samples) | *A* Ore Body (91 Samples) |
|---|---|---|
| Iron | 60.80 | 61.80 |
| Phosphorus | 0.017 | 0.023 |
| Silica | 3.42 | 2.76 |
| Manganese | 0.19 | 0.22 |
| Alumina | 0.72 | 0.90 |
| Sulphur | 0.039 | 0.038 |

Dried at 212°F.

The individual samples from which the composites were taken were mostly 5 ft. long. As far as may be judged before penetrating the ore in mine openings, the moisture content of ore from the *B* ore body will be 7 per cent, which gives a natural iron content of 56.54 per cent. For the *A* ore body, we take a moisture of 6 per cent; the natural iron would thus be 58.09 per cent.

The low silica content of the ore will render it desirable for mixing in view of the increasing average silica content in the ores now being produced in the Lake Superior Region.

A complete analysis of composite core and sludge from Hole No. 27, footage 1697 to 1775, is given in Table 3.

TABLE 3.—*Complete Analysis of Typical Ore*[a]
DRIED ORE

| | PER CENT | | PER CENT |
|---|---|---|---|
| Iron | 62.55 | Sulphur | 0.004 |
| Phosphorus | 0.017 | Lime | 0.32 |
| Silica | 3.18 | Chromium | 0.008 |
| Manganese | 0.07 | Vanadium | 0.027 |
| Alumina | 1.10 | Arsenic | 0.073 |
| Titanium | 0.09 | Zinc | None |
| Magnesia | 0.27 | Lead | None |
| Nickel | 0.008 | Copper | 0.009 |
| Cobalt | 0.003 | Loss on ignition | 5.34 |

[a] Crowell and Murray, Analysts, Cleveland, Ohio.

### MINERALOGY

#### *Texture or "Structure"*

As shown by the drilling, and by the float ore found abundantly on Float-ore Island and in various places along the south shore

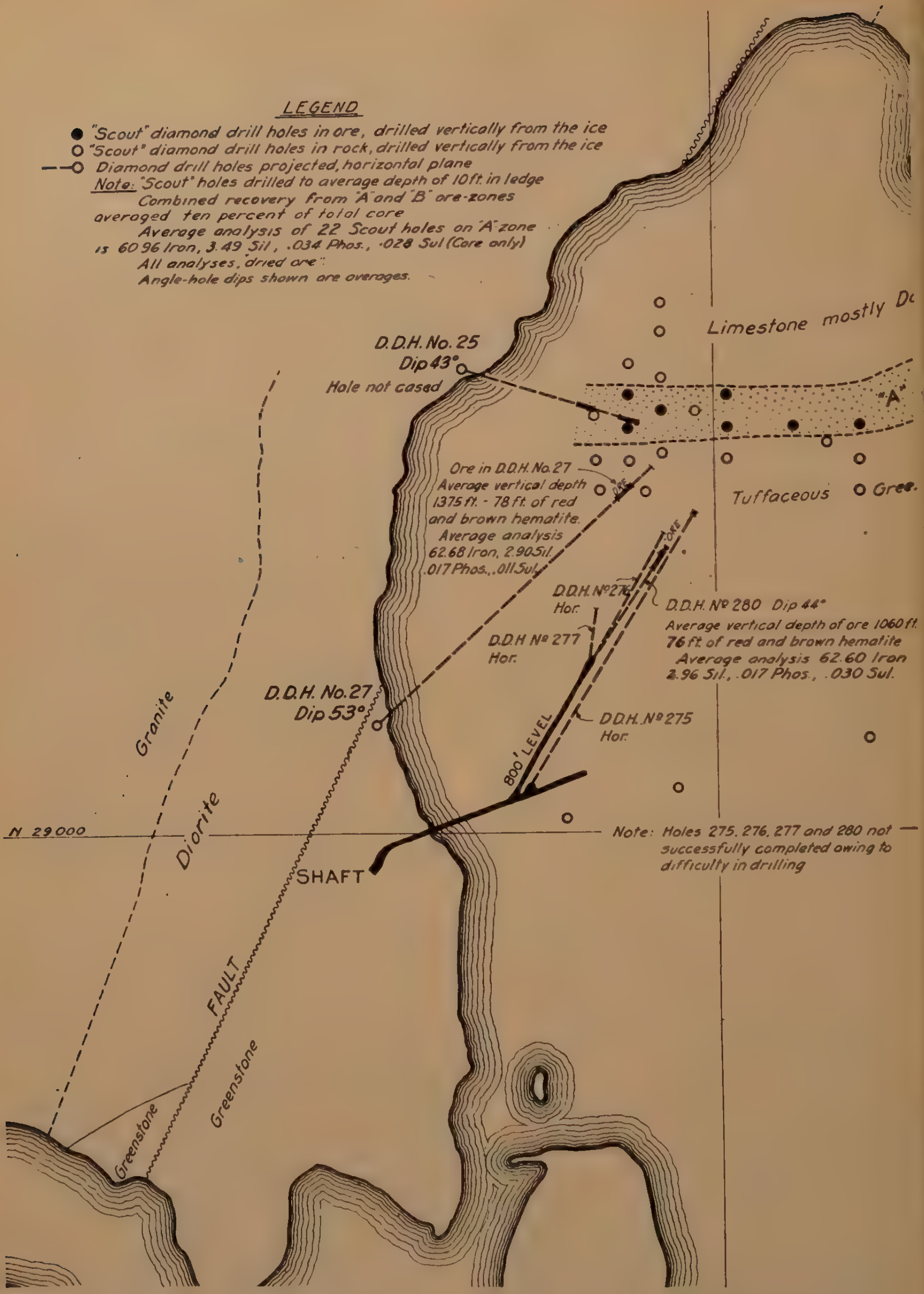
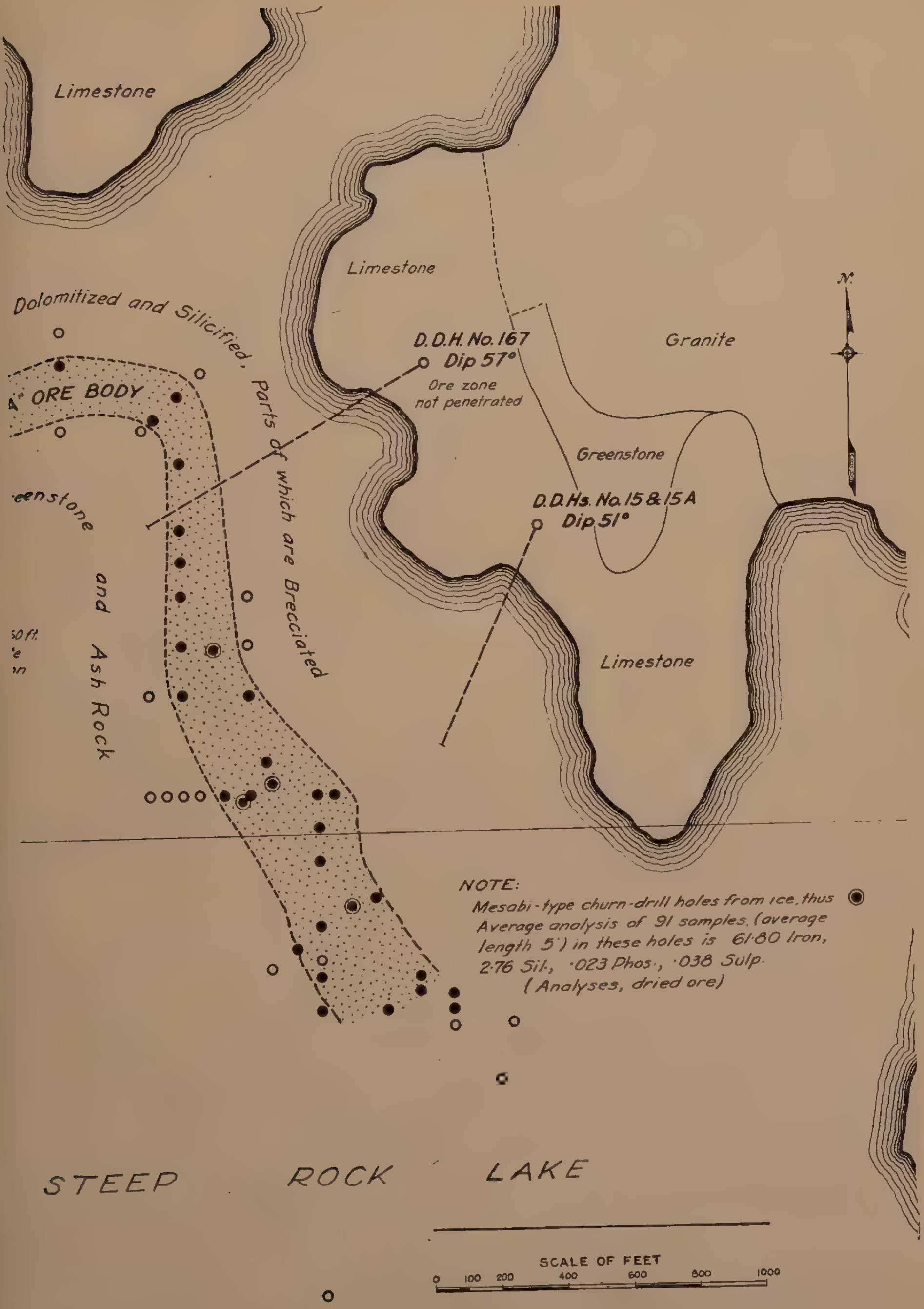

LEGEND
● "Scout" diamond drill holes in ore, drilled vertically from the ice
○ "Scout" diamond drill holes in rock, drilled vertically from the ice
--○ Diamond drill holes projected, horizontal plane
Note: "Scout" holes drilled to average depth of 10 ft. in ledge
Combined recovery from "A" and "B" ore-zones averaged ten percent of total core
Average analysis of 22 Scout holes on "A" zone is 60 96 Iron, 3.49 Sil, .034 Phos., .028 Sul (Core only)
All analyses, "dried ore".
Angle-hole dips shown are averages.
D.D.H. No. 25
Dip 43°
Hole not cased
Limestone mostly Do
"A"
Ore in D.D.H. No. 27
Average vertical depth 1375 ft. - 78 ft. of red and brown hematite.
Average analysis 62.68 Iron, 2.90 Sil. .017 Phos., .011 Sul.
ORE
Tuffaceous
Gree.
ORE
D.D.H. Nº 276
Hor.
D.D.H. Nº 280 Dip 44°
Average vertical depth of ore 1060 ft.
76 ft. of red and brown hematite
Average analysis 62.60 Iron 2.96 Sil., .017 Phos., .030 Sul.
D.D.H Nº 277
Hor.
D.D.H. No. 27
Dip 53°
D.D.H. Nº 275
Hor.
Granite
800' LEVEL
Diorite
N 29000
Note: Holes 275, 276, 277 and 280 not successfully completed owing to difficulty in drilling
SHAFT
FAULT
Greenstone
Greenstone

FIG. 2.—*A* ORE ZONE



NOTE:
Mesabi-type churn-drill holes from ice, thus ◉
Average analysis of 91 samples, (average length 5') in these holes is 61·80 Iron, 2·76 Sil., ·023 Phos., ·038 Sulp.
(Analyses, dried ore)

Steep Rock Lake.

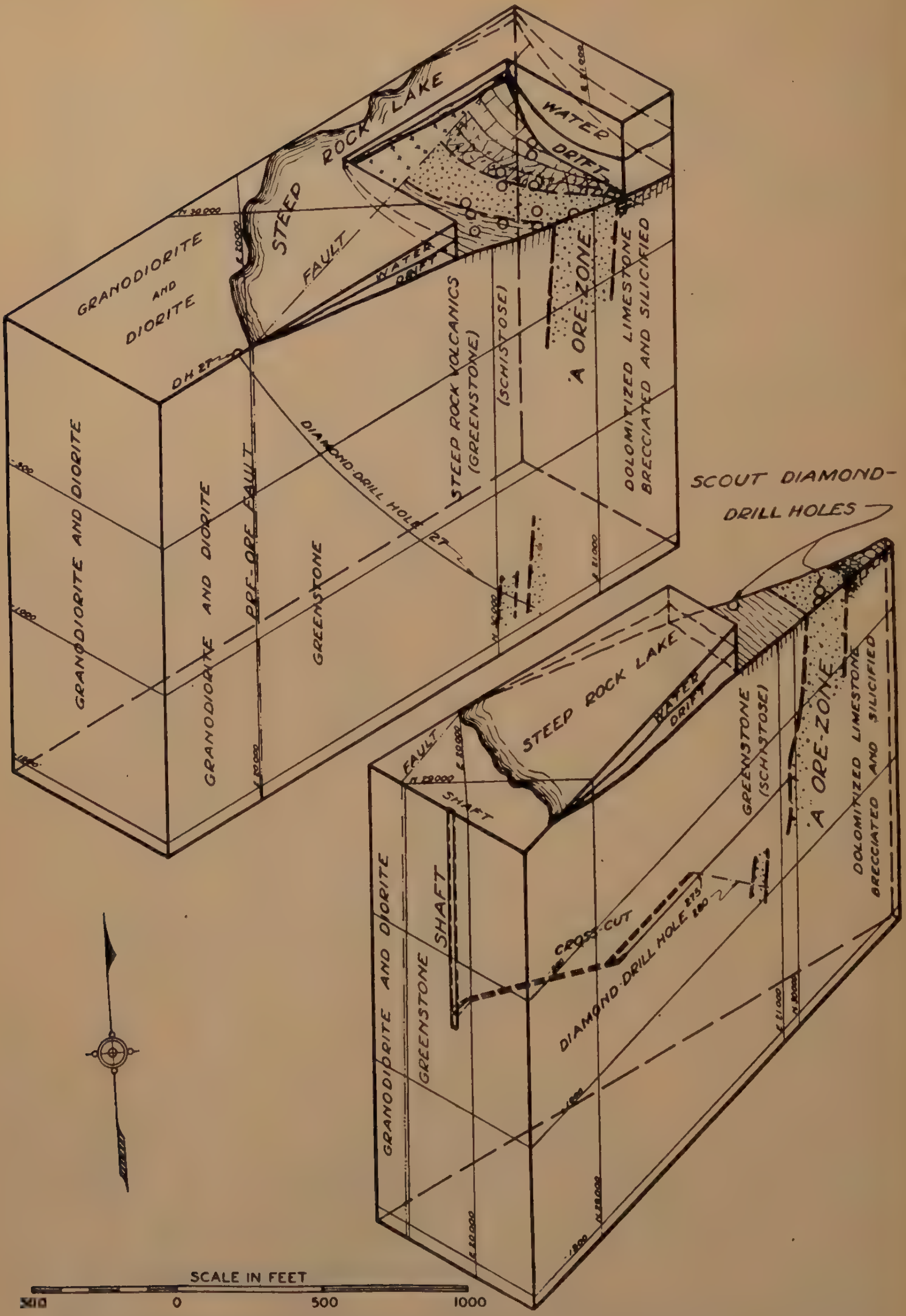


FIG. 3.—ISOMETRIC SKETCH SHOWING EXPLORATION AT WEST END OF *A* ORE ZONE.

of Steep Rock Lake, the ore is composed of hard, dense, red hematite and brown hematite (crystalline goethite). These two minerals are intimately mingled in about equal proportions. The ore is shot through with disconnected vugs varying in size from microscopic dimensions to ½ in. in diameter, occasionally attaining dimensions measurable in feet. The omnipresence of the vugs throughout the ore bodies permits the ready use of the chopping bit in Mesabi-type churn-drill practice.

The vugs are most often lined with minute crystals of hematite, sometimes specular, sometimes arranged in mammillary forms. A small proportion of the vugs contains well-formed crystals of quartz and carbonate. Some 25 per cent or more of the ore contains a smaller number of large vugs and is designated "premium lump ore" in the trade. As a whole, the "structure" of the ore is such that it would be classified as "Old Range" Bessemer. The presence of the vugs permits the ready penetration of gases in the blast furnace and in the open hearth. In fact, the vugs may have been formed originally by gaseous or vaporous emanations, which operated during the early phases of the replacement process that formed the ores.

The larger part of the ore so far drilled occurs in the form of a replaced and cemented breccia, the angular fragments of which vary for the most part from ½ to ¾ in. on a side. Most often, this breccia is found along the footwalls in the northern part of the ore zones, where it lies in contact with brecciated, dolomitized and silicified limestone of the Steep Rock series. The hanging walls of the ore zones are greenstone. Out in the hanging walls of the main ore zones there appear to be distinct lenses of ore enclosed in the greenstone (drill holes No. 27 and No. 280). The ore contained in these separate lenses is more massive and schistose than along the footwall and is dominantly red hematite. Instead of showing relict forms characteristic of breccia, it shows forms characteristic of the greenstone.

### *Microscopic Studies*

As described by G. M. Schwartz (in a private report) a typical specimen of the breccia ore from the *A* ore body has the following microscopic characteristics, as determined from a polished surface:

"This is a most unusual material. The dense, compact fragments are massive goethite. The cement is botryoidal hematite with vugs in which quartz and other gangue minerals occur. Veinlets of hematite cut the goethite and rounded remnants of goethite remain in the larger areas of hematite. Polarized light shows that the botryoidal hematite is highly crystalline; that is, equivalent to specularite and presumably high temperature in origin. A few quartz crystals occur in some of the porous areas and are also probably of late, high-temperature origin. There does not seem to be any evidence bearing on the origin of the goethite fragments."

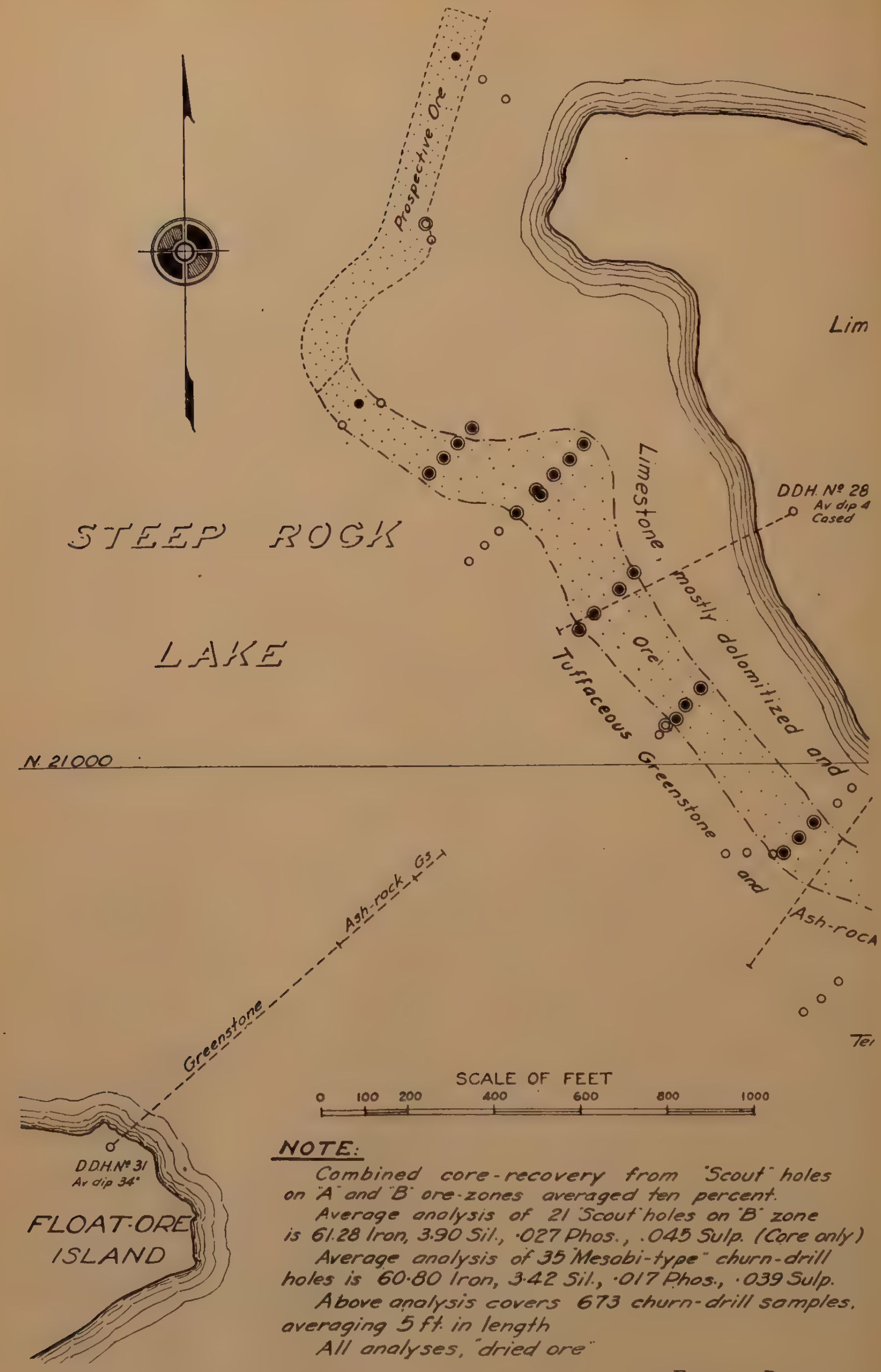
### *Thin Section Description*

A typical specimen of the hanging-wall greenstone ore material from drill hole No. 27 is also described by Schwartz, as follows:

"The thin section shows a complicated mass of fine-grained minerals with much hematite masking the characteristics of the rock as a whole. Remnants, of what appear to have been the original rock (greenstone), consist mainly of chlorite and epidote, which would be expected in the case of a much altered greenstone.

"At places the hematite is penetrated by a network of calcite veinlets which grade into a matrix for the hematite. The carbonate is clearly later than the hematite. Locally the iron oxide is goethite and needlelike crystals may be observed penetrating the remnants of the greenstone.

"The slide seems to show a clear case of replacement of greenstone by hematite, goethite and calcite."



NOTE:

Combined core-recovery from "Scout" holes on "A" and "B" ore-zones averaged ten percent.

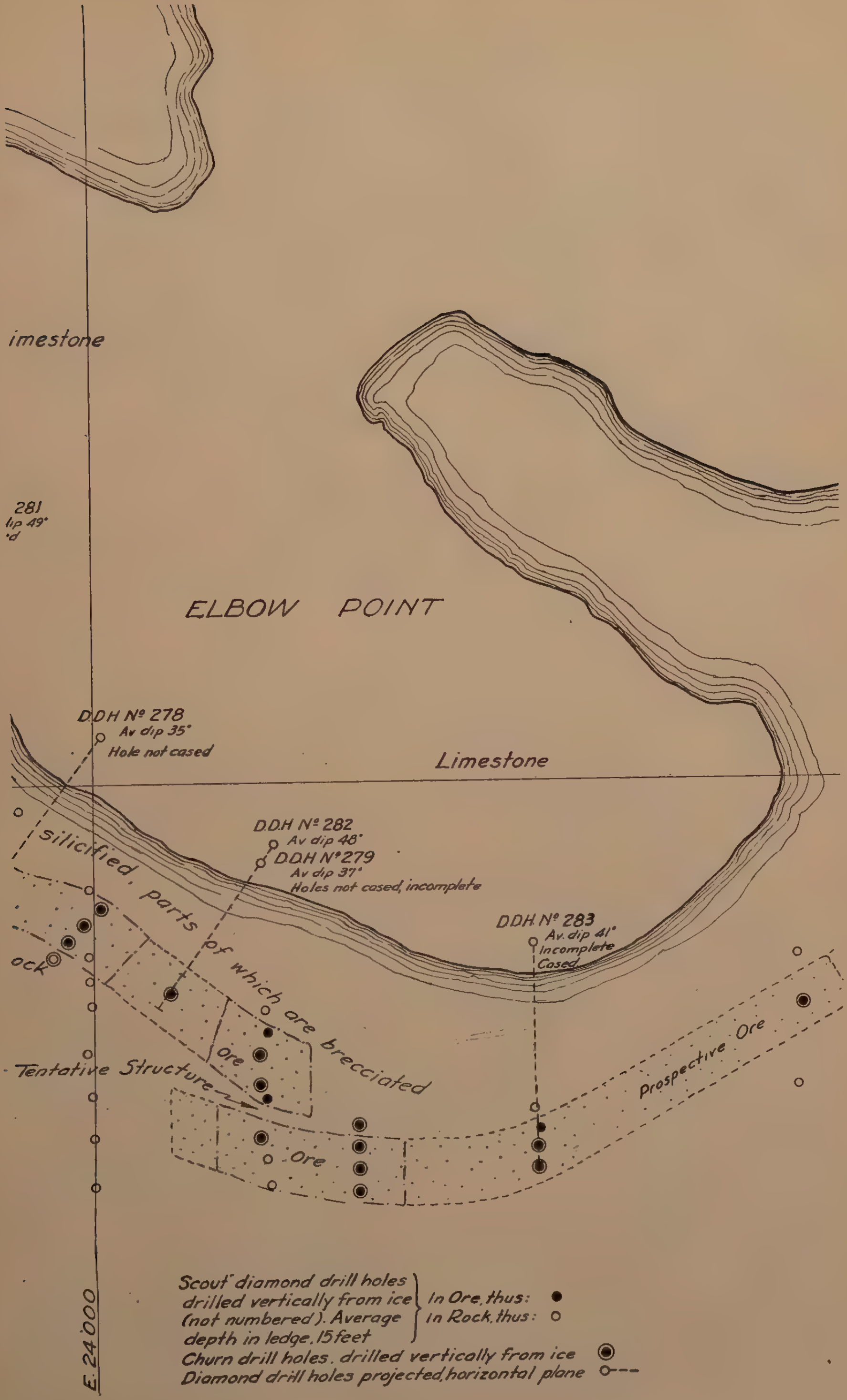
Average analysis of 21 "Scout" holes on "B" zone is 61.28 Iron, 3.90 Sil., ·027 Phos., .045 Sulp. (Core only)

Average analysis of 35 "Mesabi-type" churn-drill holes is 60·80 Iron, 3·42 Sil., ·017 Phos., ·039 Sulp.

Above analysis covers 673 churn-drill samples, averaging 5 ft. in length

All analyses, "dried ore"

FIG. 4.—*B* ORE ZONE,


imestone
281
lip 49°
'd
ELBOW POINT
D.D.H. Nº 278
Av dip 35°
Hole not cased
Limestone
D.D.H. Nº 282
Av dip 48°
D.D.H. Nº 279
Av dip 37°
Holes not cased, incomplete
D.D.H. Nº 283
Av. dip 41°
Incomplete
Cased
silicified, parts of which are brecciated
ock
Ore
Tentative Structure
Ore
Prospective Ore
"Scout" diamond drill holes drilled vertically from ice (not numbered). Average depth in ledge, 15 feet
In Ore, thus: ●
In Rock, thus: ○
Churn drill holes, drilled vertically from ice ◉
Diamond drill holes projected, horizontal plane ○---
E. 24,000

STEEP ROCK LAKE.

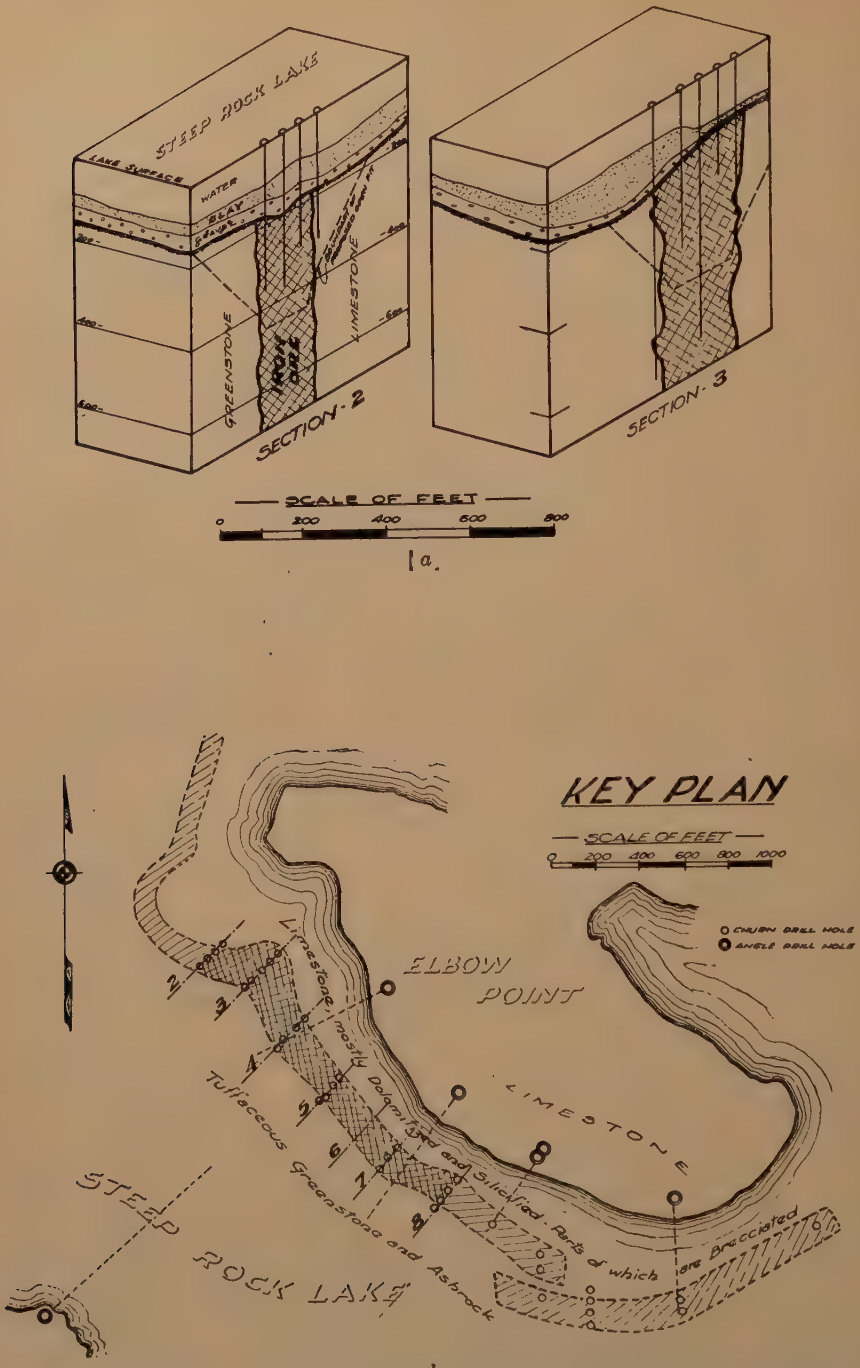
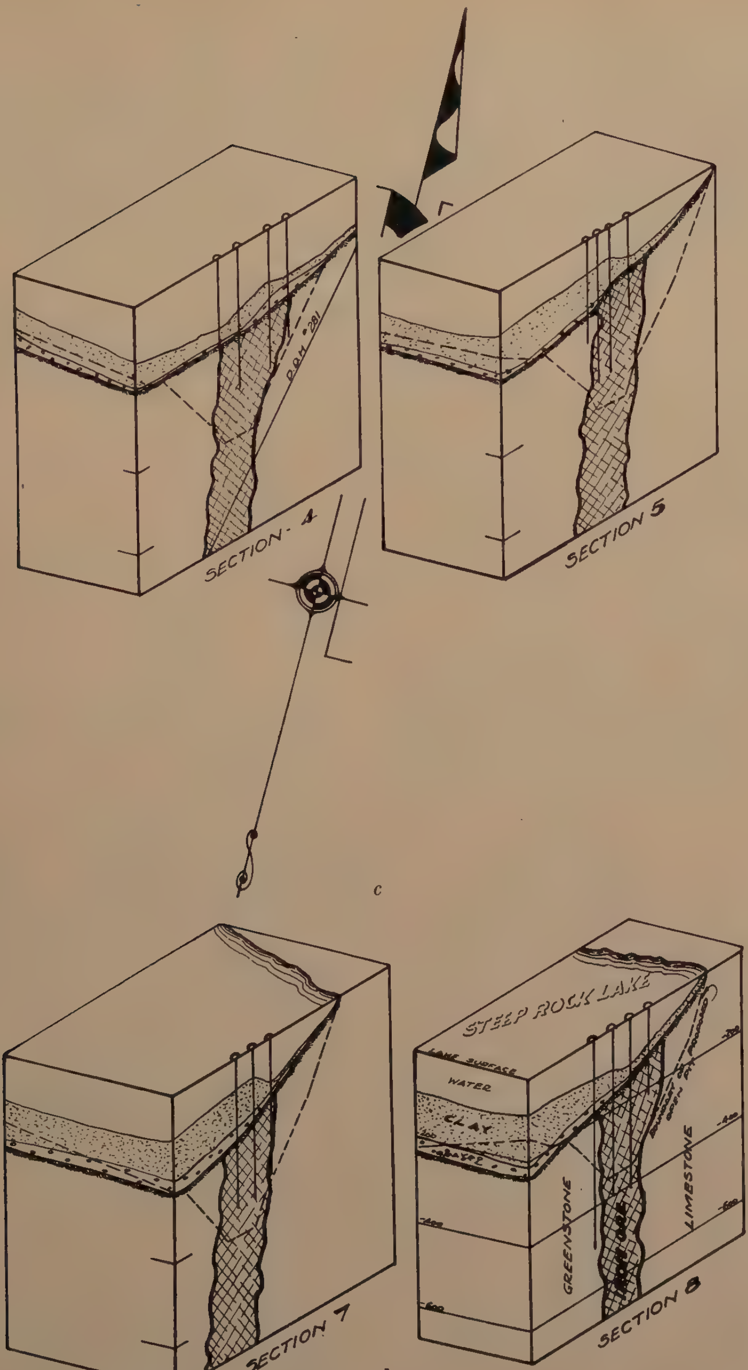


FIG. 5.—ISOMETRIC SKETCH SHOWING EXPLORATION



*c*

*d*

IN CENTRAL PORTION OF *B* ORE ZONE.

### SIZE OF ORE BODIES

By the work to date, the ore zones have now been shown to be comparable in areal extent with known ore bodies of like nature on the Vermilion Range in Minnesota and the Marquette Range of Michigan, as shown in the various figures accompanying this paper and in Table 4. The greatest vertical depth of ore now disclosed is in hole No. 27 at 1210 ft. below bedrock at the bottom of the lake (1450 ft. below lake level). Considerable portions of the greenstone-limestone contact remain unexplored, as in Southeast Arm, Falls Bay, and West Arm.

TABLE 4.—*Measurements Relating to Ore Zones as Partly Marked Out by Drilling up to April* 1942

| Ore Zones | Length Explored, Ft. | Width Explored, Ft. | | Depth of Water, Ft. | | Depth of Overburden, Gravel and Clay, Ft. | |
|---|---|---|---|---|---|---|---|
| | | From | To | From | To | From | To |
| *A* | 3,100 | 175 | 200 | 120 | 160 | 60 | 235 |
| *B* | 4,800<br>(Open at both ends) | 85 | 225 | 50 | 180 | 40 | 310 |
| *C* | 800<br>(Open at both ends) | ? | 200 | 210 | 265 | 85 | 180 |

### TONNAGE

An adequate presentation of the nature of ore bodies should include some mention of the tonnages now disclosed. For the purpose of estimating quantities, in accordance with definitions that meet with the general approval of mining engineers, the ore so far proved and indicated has been classified under three heads; proved ore, probable ore and prospective ore.

Because considerable quantities of ore have been sampled accurately in close detail in the *B* ore body, it is now possible to estimate that 9,157,444 tons have thus far been proved, 8,400,600 tons of which can be mined in an open pit. On the *B* and *A* ore bodies together, 14,377,709 tons of probable ore are estimated, in addition to 1,525,320 tons of proved ore in the *A* ore body.*

The quantitative confirmation that has been given to the geological hypothesis presented in this paper, and upon which the last two winters' drilling has been based, gives warrant to the belief that the ultimate amount of prospective ore that may be developed in the portions of the ore zones as yet undrilled will be quite in accordance with similar quantities found in other deposits of the deep-seated type in the Lake Superior Region.

### MINING OPERATIONS FEASIBLE AND PRACTICABLE

Engineering studies show that the waters of the Seine River flowing into the northeast corner of Steep Rock Lake can be diverted to the north and west (Fig. 6); also that Steep Rock Lake can be pumped out without excessive cost. After this, the upper parts of the *A* and *B* ore zones can be mined economically and profitably from open pits and underground. It is not the purpose of this paper to discuss these engineering features, but we may express our opinion that there will be no deterrents, from a geological standpoint, that will entail any unusual hazards in this operation. As will appear from our discussion of the manner of

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* These estimates were made by William H. Crago, Consulting Mining Engineer, to whom the authors are also indebted for many valuable suggestions during the preparation of this paper.

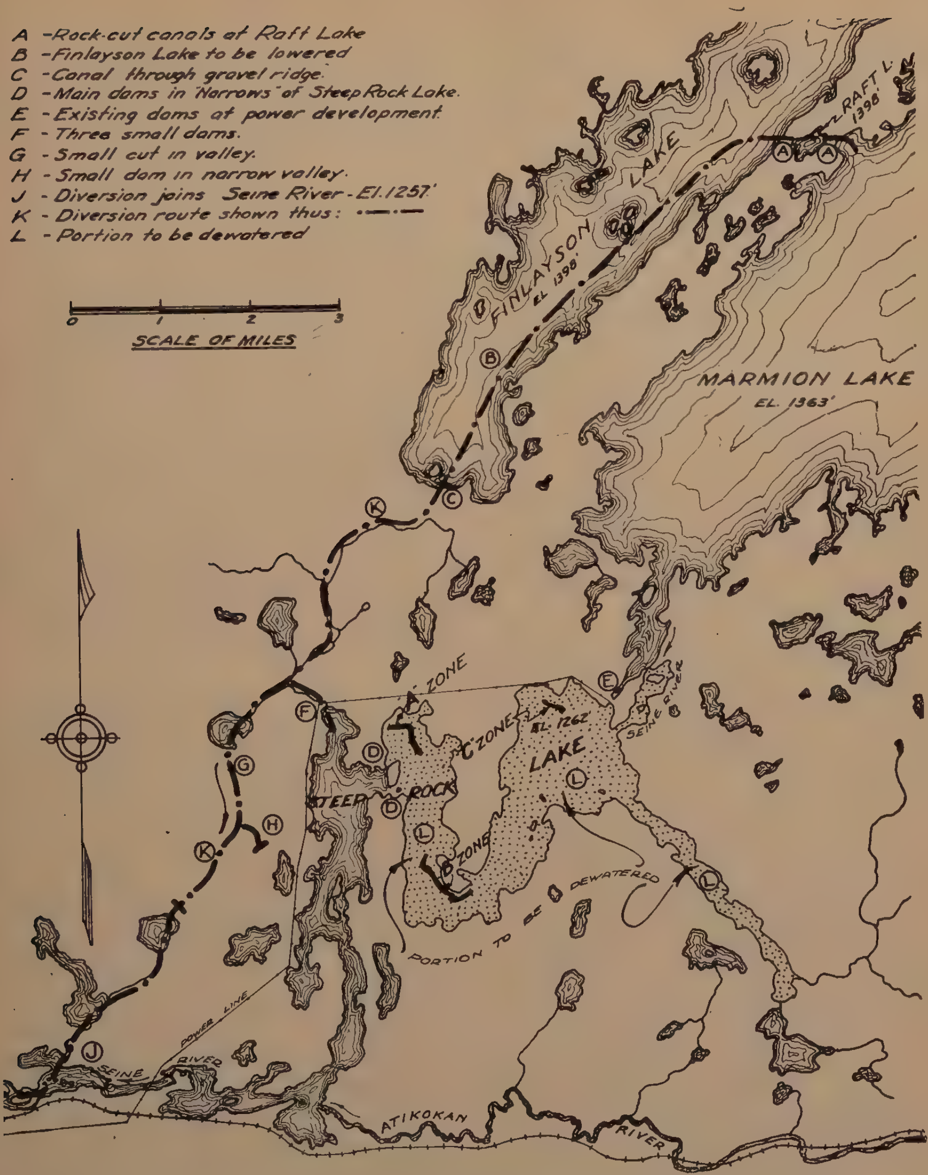
A -Rock-cut canals at Raft Lake
B -Finlayson Lake to be lowered
C -Canal through gravel ridge.
D -Main dams in "Narrows" of Steep Rock Lake.
E -Existing dams at power development
F -Three small dams.
G -Small cut in valley.
H -Small dam in narrow valley.
J -Diversion joins Seine River - El. 1257'
K -Diversion route shown thus: •—•—
L -Portion to be dewatered
0 1 2 3
SCALE OF MILES
FINLAYSON LAKE
EL. 1398'
RAFT L.
1398'
MARMION LAKE
EL. 1363'
"A" ZONE
"C" ZONE
"B" ZONE
STEEP ROCK LAKE
EL. 1262'
SEINE RIVER
PORTION TO BE DEWATERED
POWER LINE
SEINE RIVER
ATIKOKAN RIVER
A
A
B
C
D
D
E
F
G
H
J
K
K
L
L
L
L

Fig. 6.—Plan of Steep Rock Lake area, showing diversion scheme.

origin of the ores, which are of a deep-seated type, we believe that subsequent to the exhaustion of the open pits, the bulk of the ore will be won from underground mines.

## Geological Basis for Exploration and Mining

Such, in outline is the nature of the ore deposits at Steep Rock Lake, given as a basis for the real subject of this paper; i.e., an analysis of the geologic process by which the ore deposits were formed. An adequate explanation of this process is of the utmost importance, for the reason that opinions in that regard go far toward determining conceptions that may logically be entertained as to depth, volume and grade of the ores, as well as the methods by which they may be best explored and mined.

We believe that the deposits were formed as the result of replacement by iron oxides through the agency of hydrothermal solutions and the accompanying action of vapors under pressure. This process, in the culmination of which clean iron oxides were formed, has gone quite fully to completion except for minor remnants. This is shown by the notable uniformity of the several constituents in the ore throughout.

Since the Steep Rock district lies within the Lake Superior Region, as the work proceeded we thought it of consequence to determine first, as far as practicable, a correlation with the rock sequences in other iron-bearing districts in the general area. We sought also a comparison with the geologic processes that have operated elsewhere, hoping in this way to enlarge our understanding of the exploratory problem.

## General Geology

Because of the complexity of the problem, there has been much diversity of opinion among geologists as to the age of the rocks in the Steep Rock basin. The unique occurrence of limestones there, situated as these rocks are, within a broad area of dominantly old igneous rocks, has drawn the attention of pre-Cambrian geologists since 1890 and before. These limestone beds obviously have been preserved from erosion by their position within a limited area of subsidence (Fig. 7).

Based on the intensive field studies we have made, and supplemented by the observations of many other investigators, especially T. L. Tanton, Table 5 broadly outlines the geological succession.

Table 5.—*Formations*[a]

| | |
|---|---|
| Post-Steeprock | Acid and basic minor intrusives. |
| *Intrusive Contact* | |
| Steeprock Volcanic Formation (Formerly classified as Keewatin) | Intermediate and basic lavas with intercalated sediments. Intermediate and basic tuffs and agglomerates genetically related to the above lavas. |
| *Erosional Unconformity* | |
| Steeprock Sedimentary Formation | Limestone with minor amounts of clastic sediments. |
| *Erosional Unconformity* | |
| Pre-Steeprock | Older gray granite and included greenstone. |

[a] The Steeprock volcanics are here differentiated from the Keewatin and classified as younger than the Steeprock sediments.

The Seine or Couchiching formation to the south, which has been subject to much controversy, is believed to have no relation to the ore deposits and is not discussed here.

### *Correlation*

The rock formations at Steep Rock Lake correspond very closely, lithologically and stratigraphically, to similar formations on the Marquette and Menominee Ranges of Michigan, except that in the Steep Rock area the characteristic banded iron formation is lacking. It may be beyond our purview at this time to propose a precise and complete correlation between these areas, but the striking similarity warrants more than the drawing of a cursory analogy. The stratigraphic succession at Steep Rock Lake, of granite, overlain successively by conglomerate, limestone and volcanics, bears a close resemblance to the succession of Algoman granite, Sturgeon quartzite, Randville or Kona dolomite, and the Hemlock greenstone of the Michigan ranges. On the basis of this correlation, the granite underlying the Steep Rock series is of Algoman age and the overlying rocks would thus be chronologically allied to the Lower Huronian of Michigan. This correlation is less conjectural than those previously proposed.[1–5] Limestone with similar distinctive concretionary forms is found interlayered with banded iron formation and greenstone flow rocks on the Nastapoka and Belcher Islands of the east coast of Hudson Bay. The age of these rocks is not definitely known but earlier study of the succession on the Nastapoka Islands by one of the authors reveals many points in common that furnish a reasonable basis for correlation with the succession at Steep Rock.

The oldest rocks in the Steep Rock basin are represented by remnant inclusions of highly altered basic schists that may be Keewatin. These inclusions are found within gray schistose granite. These rocks are confined to the north and east shores of the lake. The age of the granite has long been in question. T. L. Tanton* classifies it as Algoman. He maintains that this granite underlying the sediments is younger than the Steep Rock sediments and is intrusive into it. While we agree that the granite is in all probability of Algoman age, we cannot fully subscribe to his observations as to the intrusive nature of the contacts.

In contact with this granite, and apparently overlying it, is a succession of clastic and chemical sediments, approximately 1300 ft. thick, known as the Steeprock Sedimentary Formation. The basal member is a conglomerate containing pebbles of the underlying granite, also pebbles of Keewatin greenstone and of quartz. The limestone member that makes up the bulk of this formation overlies the clastics in places and is in direct contact with the granite at other places. When direct contact occurs, the usually expected contact-metamorphic minerals such as lime silicates are totally absent from the limestone-granite contact zone; thus we believe that we are justified in stating that the Steeprock series is younger than the underlying granite.

The deposition and consolidation of the Steeprock sediments were followed by emergence and no doubt by a considerable period of erosion, probably comparable in extent to that between the Lower and Middle Huronian periods. Finally volcanism ensued on a large scale. During the early part of the volcanic disturbance, a succession of tuffs and agglomerates, intermediate to basic in composition, were deposited. A basic tuff, known locally as "ashrock," is distinguished by its peculiar mineralogical character. Concretionary balls of pyrite and inclusions of limestone, old schist and banded cherts, both rounded and subangular, are prominently displayed near the base of the "ashrock." The presence of these inclusions, taken with the disconnected distribution of the limestone, has been interpreted by us as evidence of an unconformity between the underlying limestone and the "ashrock." The thought is that a considerable space of time separated the consolidation of the limestone and the disposition of the "ashrock." During this time, erosion produced many boulders and fragments of the earlier rocks, which subsequently were enveloped by the newly deposited volcanics. Unfortunately, shearing has destroyed most of the structure in the basic tuff, and the postulated unconformable contact with the limestone is nowhere exposed.

Lavas of a similar composition succeed the tuffs. These lavas are unaltered and appear to be relatively young. Many are amygdaloidal and have fresh, well-formed pillows, fragmental tops and chilled borders. Intercalated sediments, thought to be younger than the Steeprock sediments, appear intermittently throughout the lava series.

Intrusive into the Steeprock sediments and volcanics are two groups of dikes, stocks and sills, one acid in character, the other basic. The former, and on the whole earlier, group is represented by felsite and acid porphyry dikes and small stocks. Later diorite, gabbro and peridotite dikes and sills comprise the larger part of the intrusive aggregate in the map area. These intrusives are found more plentifully in this vicinity than in adjoining areas and occur largely within a basin of depression, as mapped in Fig. 7. Along the north edge of this basin, preceding the development of ore, there was an early introduction of silica and carbonates that came from a source believed to be genetically related to the acid dikes and sills. Also, we believe, during the course of the same process of differentiation, the basic iron-bearing in-

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* Personal communication.

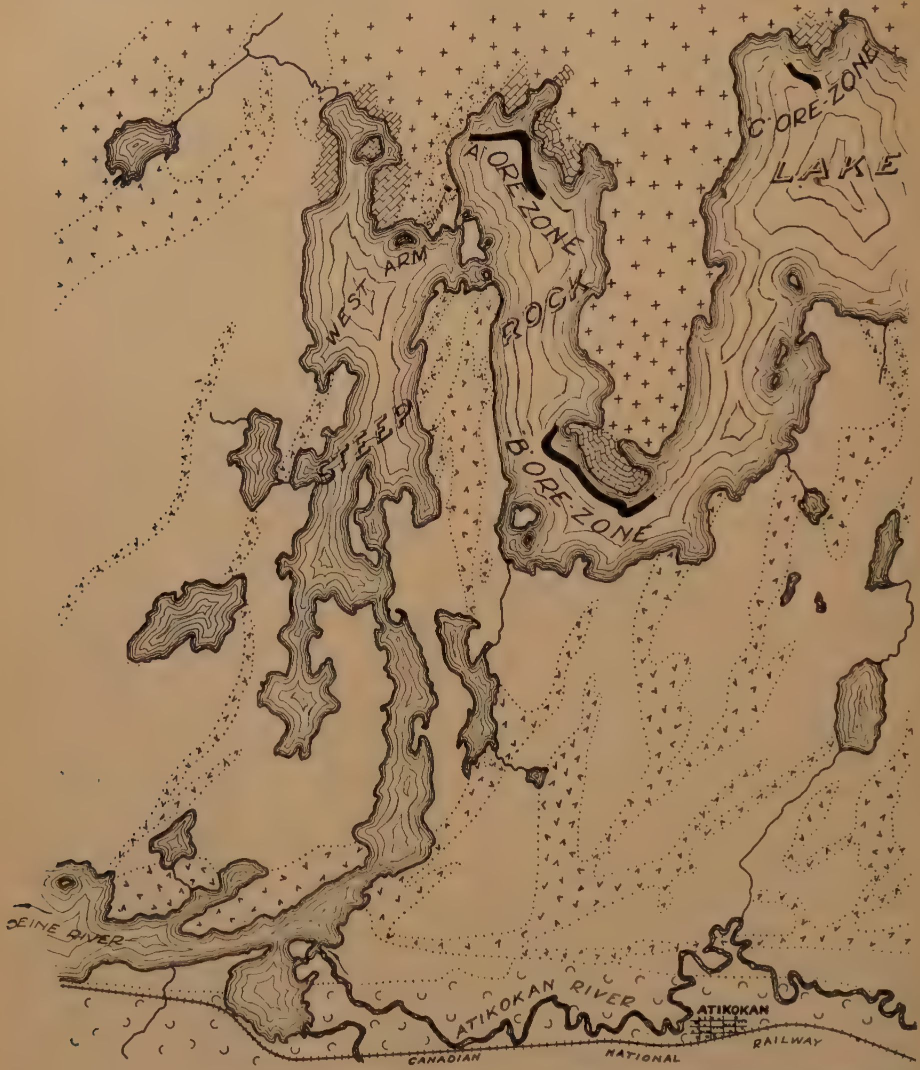
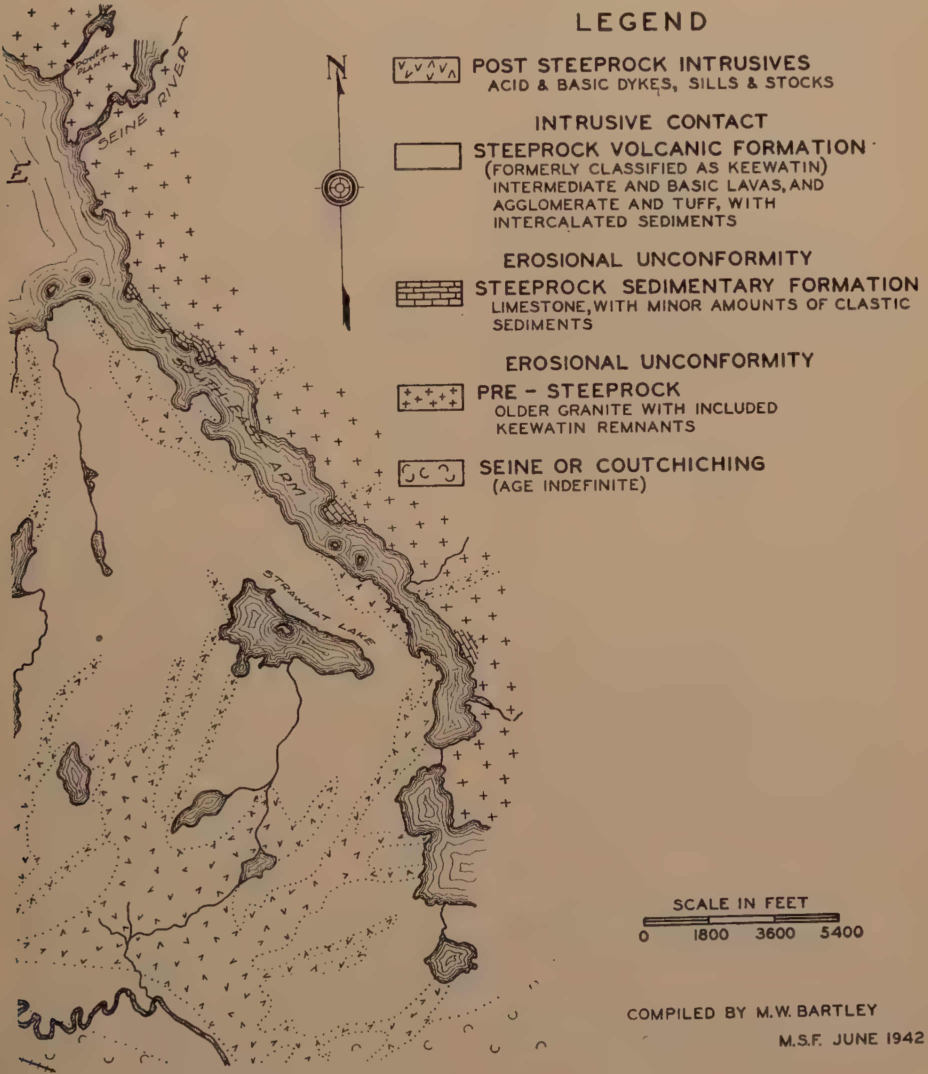


FIG. 7.—GENERALIZED GEOLOGICAL

trusives that contain abundant ilmenite and magnetite were associated with and closely preceded the emanations that brought in the hematite, as will be discussed more fully in a subsequent section. These emanations caused replacement of the previously deposited silica and carbonate and also certain parts of the wall rock that were favorably disposed to replacement. The final expression of magmatic action resulted in post-ore quartz, carbonate stringers and vug fillings.


LEGEND
POST STEEPROCK INTRUSIVES
ACID & BASIC DYKES, SILLS & STOCKS
INTRUSIVE CONTACT
STEEPROCK VOLCANIC FORMATION
(FORMERLY CLASSIFIED AS KEEWATIN)
INTERMEDIATE AND BASIC LAVAS, AND AGGLOMERATE AND TUFF, WITH INTERCALATED SEDIMENTS
EROSIONAL UNCONFORMITY
STEEPROCK SEDIMENTARY FORMATION
LIMESTONE, WITH MINOR AMOUNTS OF CLASTIC SEDIMENTS
EROSIONAL UNCONFORMITY
PRE - STEEPROCK
OLDER GRANITE WITH INCLUDED KEEWATIN REMNANTS
SEINE OR COUTCHICHING
(AGE INDEFINITE)
N
POWER PLANT
SEINE RIVER
E
ARM
STRAWHAT LAKE
SCALE IN FEET
0
1800
3600
5400
COMPILED BY M.W. BARTLEY
M.S.F. JUNE 1942

MAP OF STEEP ROCK LAKE AREA.

The host rocks of the ore, we believe, are of Lower Huronian and Early Middle Huronian age. The entrance of the ore very likely accompanied the mountain building that took place in the northern part of the Lake Superior Region. In all probability this occurred at about the time when the bedded iron formations of the Mesabi Range were being laid down in the seas to the south. During the same period of mountain building, it may well be that hydrothermal replacements were also formed in the folded and brecciated, lean Keewatin iron formations on the Vermilion Range.

## DEPOSITS ARE HYDROTHERMAL REPLACEMENTS

### *Earlier Explanations*

Various explanations of the origin of the ores at Steep Rock Lake have been suggested by those who have studied them.

First, there is the explanation that these deposits have been formed by the secondary action of cold surficial waters upon banded iron formation. Our studies show a very limited thickness or areal extent of rocks in the district that could possibly be construed as banded iron formation, insufficient in our opinion to have supplied the great quantity of iron that is present in the ore zones.

Second, for the same reasons, it does not seem possible that the deposits could have resulted from the secondary, surficial alteration of the limited amounts of iron carbonate or ferrodolomite that are present in the area.

Third, we see no indication that the deposits are the result of the deep weathering of iron sulphides. The ores are dense and show no evidence of the extreme porosity that characterizes iron ores developed in this way. There is no reason to believe that there has been sufficient metamorphism of the ores, subsequent to their development, to account for their present density. The only sulphide so far found is pyrite. This occurs in isolated and confined areas in the hanging wall of the main ore zones and is found there interlocked with the hematite as though it were formed contemporaneously with the hematite and in some instances slightly later.

Since none of these explanations of the origin of the ores seem tenable, we have searched for another explanation.* In the discussion of the age of their deposition, we have related them to the main time of development of the large iron formations in the Middle Huronian period of the Lake Superior Region. While the Steep Rock ores are in the same metallogenetic province as ores in the producing mines to the south, the detailed manner of their deposition was somewhat different from the process usually assigned for the development of the iron-ore bodies of Minnesota and Michigan.

### *Primary Hydrothermal Replacements*

To our way of thinking, the Steep Rock ores were formed as primary replacements by ascending magmatic solutions. This is not an unusual geologic process, but is quite in accordance with world-wide processes that have been assigned to the genesis of valuable iron-ore deposits in many parts of the world. Such deposits, to mention a few, are found in Utah, Missouri and other western states, also in Sweden, Venezuela, southeastern Cuba and Chile. The Cumberland hematites of England have been traced to a similar source.[7] Collins, Quirke and Thomson found abundant evidence of direct replacement of igneous rocks by iron-bearing solutions in the Michipicoten district of Canada.[6]

T. L. Tanton, who has made exhaustive studies of the Steep Rock area since 1924 or earlier, is also of the opinion that the hematite iron ores of Steep Rock Lake are primary replacement deposits, formed by ascending mineralizing solutions. As early as 1925, he recognized replacement phenomena within areas in close proximity to the ore zones as now developed. Later, in a paper presented to the Royal Society of Canada in 1941,[8] he made the following statement, which is most applicable:

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* We wish to acknowledge particularly our indebtedness to W. D. Neeland, who was resident geologist at Steep Rock Iron Mines Ltd., from 1938 to 1941.

It is probable that they (the deposits) were introduced by ascending mineralizing solutions. In point of time, the mineral deposits are more closely related to the ashrock and late intrusive rocks of the area than to the older sediments and greenstones. It is considered probable that they were introduced in successive surges of variable composition during the time of late igneous activity.

In our opinion the ore-depositional processes that took place at Steep Rock Lake were somewhat as follows:

### *Folding and Brecciation*

In the later stages of the Steep Rock volcanism, accompanying and subsequent to the intrusion of the closely related intrusives, there came a foundering of the basin against the older Algoman granite buttresses to the north. This tilted the Steep Rock sediments and overlying volcanics to variable angles, averaging about 75° to 80° southward. Contemporaneously with this foundering, there came pressures from the west that folded and crumpled these rocks against another buttress of granite that lies east of the Southeast Arm of Steep Rock Lake. Similar deformation occurred along the Atikokan Range immediately to the east, as mapped by Hawley.[9] These movements, the foundering principally, deformed the rocks along the limestone-greenstone, unconformable contact, resulting in the development of open breccias.

Preceding and accompanying this foundering, which greatly disturbed the Steep Rock volcanics in the southern part of the area, there came large-scale intrusions of acid and basic dikes, stocks and sills, which penetrated these volcanics. By far the larger part of these masses were basic in composition. Following immediately upon the intrusions, there came emanations in large volume that dolomitized and silicified the previously formed breccias and schists along the limestone-greenstone contact. All of the limestone remote from this contact was not dolomitized, which shows that the process was selective and had a spatial relation to the zone of deformation.

The uprising injections, which, in all likelihood, came from the same reservoirs that yielded the intrusives, were essentially continuous in their process of mineralization, producing different effects according to their changing composition[10] and the character of the host rocks. At the beginning, these solutions probably contained much silica and carbonates of lime and magnesia but only minor amounts of iron carbonate and little sulphur. This is evidenced by the remnants of their effects now to be seen. There are siliceous borders extending out from both walls of the ore bodies. Minor siliceous remnants that appear to have been overwhelmed by later solutions of different composition occur within the ore zones.

During the dolomitization and silicification, slight movements disturbed and rebrecciated these newly formed zones. This new disturbance provided the channels and openings for the transportation and lodgment of the uprising iron-rich portion of the major emanation. Evidence for this and the replacement phenomena are to be found in the "float" and in the drill core.

The invading solutions increased in aggregate volume and quantity of iron, as evidenced by the facts shown in the field and in the drill holes. Owing to a release of oxygen by the replacement of finely comminuted limestone, which will be discussed in a subsequent paragraph, more intense oxidation of the iron-bearing solutions occurred and a heavy deposition of iron as hematite resulted. This reaction may have been stimulated by catalytic action due to minor amounts of hematitic oxide remaining from the previous stage of deposition.

### *Oxidizing Effect of the Brecciated Limestone*

Because ore bodies are found on the brecciated contact of limestone, or close to large bodies of brecciated limestone—

and as far as now known only there—the limestone probably had a determining effect on the nature of the ore deposition. Its presence may have caused deposition of hematite rather than of the lower oxide or sulphides. The influence of limestone on the deposition of iron minerals has been considered by B. S. Butler, notably in a paper[11] in 1923, "A Suggested Explanation of the High Ferric Content of Limestone Contact Zones." One great effect in this particular instance may have been the release of oxygen during the decomposition of finely broken limestone. During this procedure there would have been a breakdown of the carbonate into $CO_2$, CO and O and a further change to C and $O_2$.[12] This process acting in a zone of extreme brecciation has resulted in the development of brown and red hematites only in the main ore bodies, rather than magnetite or a mixture of hematite, magnetite and sulphides.

Also, the deposition of pyrite in certain confined portions of the hanging wall tends to confirm this observation. Excess oxygen from the limestone on the footwall may have been consumed in the replacement process that went on in the central part of the ore zone and thus did not reach confined areas. Hence the iron in solutions that penetrated to these places was deposited as pyrite (as discerned by Neeland). For instance, hole 280, drilled from underground, intersected 76 ft. of ore in a zone 106 ft. wide as measured along the length of the hole. This is believed to represent a hanging-wall lens. Within this ore, a 5-ft. section is vitiated by sulphur as shown in the analysis; visible pyrite appears within the hematite in this section. As another instance, in a section within the hanging wall in the southern portion of the *B* ore zone, pyrite is found in siliceous replacement. It is to be noted that similar occurrences are found in connection with producing mines on the Vermilion Range in Minnesota at the feathering edges of ore bodies.

### *Sequence of Iron Oxides*

*Deposition.*—Because the brown hematite or goethite, being hydrous, contains more oxygen in the aggregate than does the red variety, we infer that it was the first iron oxide to be formed in the ore zones, and especially is this probable since goethite occupies the central portion of the ore bodies and constitutes the replacement fragments in the breccia ores. As the excess oxygen content was consumed in the development of this hydrous, brown hematite, the supply of oxygen may have lessened slightly and a deposition of red hematite ensued. This is inferred because red hematite is found throughout the ore bodies as the cementing material and in stringers. In this connection it is important to note that in the parts of the ore zones farthest from the limestone, where the ore is enclosed in greenstone, it is in precisely those portions that the ore is most largely red hematite. This sequence is supported by the microscopic studies of G. M. Schwartz.

*Correlation with the Atikokan Range.*—In this connection, it is most important to note that large deposits of iron ore consisting of mixed pyrrhotite, magnetite and minor amounts of pyrite, are found on the Atikokan Range, 10 miles to the east of Steep Rock Lake. The best of these deposits have a content of iron, phosphorus and silica nearly identical with that contained in the ores of Steep Rock; i.e., 59.80 per cent iron,[13] 0.025 per cent phosphorus, 3.30 per cent silica, but the sulphur runs 20.40 per cent.

This striking similarity in the analyses of this ore with that of Steep Rock Lake (save in sulphur content), may be an important key to a proper understanding of the process that produced the ores at Steep Rock Lake. According to Hawley,[9] the Atikokan deposits are of the magmatic replacement type, and our own observations confirm this. They are found along a fault zone in basic schists where there are abundant basic intrusives. The faulting took place at the same time as the deformation at Steep Rock Lake and was part of the same general movement. The great difference in the two places is the *absence of limestone* on the Atikokan Range; hence we may infer that there was a lack of oxygen when the solutions came in, thus magnetite and pyrrhotite were formed rather than hematite.

*Range of Deposition.*—As revealed by the silicification and dolomitization that preceded the deposition of hematite at Steep Rock, the early stages of the solutions were probably high in silica and magnesia. At a later stage, when the invading solutions were depositing iron most extensively, they were probably undersaturated with respect to silica, since the best ore usually contains less than 3 per cent silica. As a result of a continuing injection of iron-bearing solutions, the effects of earlier depositions were no doubt overwhelmed and most of the silica and magnesia present were removed and transferred farther into the walls of the deposits, where they are now found. When total replacement was reached in the central zone and in the adjoining ore bodies, equilibrium would have been established between the solid hematite and the remaining solution, which had become less rich in iron. This residual solution probably went out into the walls; the final expression of its effects is shown by the existence of "paint rock" and the hematitic alterations that are known to extend for considerable distance with diminishing effect as the solutions passed outward.

During the same process, some of the iron-bearing solutions penetrated farther out into the greenstone hanging walls. They deposited disseminated sulphides and magnetite under the reducing conditions that prevailed in those basic rocks. A different environment caused dissimilar depositional effects; i.e., hematite was deposited in the one place, magnetite and sulphides in the other. No doubt there was some original magnetite in the volcanics, but it is our belief that much additional magnetite was introduced at the time the ore deposits were formed; in fact, Messrs. Errington, Cross* and Samuel† were led to the discovery of the *A* and *B* ore zones by drilling toward the limestone from two distinct areas of high magnetic attraction situated out in the volcanics.

### *Brecciated Appearance of Ore Due to Replacement of Original Fragments*

The ores show a brecciated texture except where the ore is believed, without doubt, to be replaced greenstone. The brecciated texture of the main zones may be accounted for most reasonably by assuming molecule by molecule replacement of original fragments. While it is very likely that some movement, subsequent to its development, took place within the ore, we believe that no great movement could have taken place, else the water of hydration in the brown hematite would have been driven off and megascopic specularite developed. Viewing the situation on a large scale, the possibility of slight movement cannot be ruled out. Such a movement, by causing limited dehydration, could have developed some of the red hematite that is found dominantly along the edges of the ore bodies.

### *Manganese Indications*

In the walls, pinkish carbonates are found, indicating the presence of manganese. Manganese oxides occur in minor quantity in all of the various types of replacements. It is not beyond the realms of reasonable probability that manganese ores may be found along the footwalls of the main ore zones, where they may be pockets of sufficient size and purity to mine. The chemistry of the manganese deposition no doubt followed closely that of the iron.

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* Julian G. Cross, Mining Engineer, a Director of Steep Rock Iron Mines Limited, one of the initiators of the exploratory work.

† W. Samuel, Consulting Mining Engineer, Steep Rock Iron Mines, Ltd., to whose acumen the Company has often been beholden.

Manganese oxides occur around the shores of the lake, interbedded with clays, but in minor quantities.

### *Subsequent Surficial Alteration*

The principal evidences of the effects of subsequent circulating cold surface waters upon the deposits of Steep Rock Lake are soft "muds" that some of the drill holes have encountered within the "paint rock." The paint-rock zones may have been primarily altered walls of the ore bodies. These paint-rock belts are adjacent to the ore zones; we believe that originally they were the product of alteration by heated solutions and have since, in some places, been converted into muds by surface waters. These mud-occupied fractures, down which the surface waters percolated, may be healed at not too great depth, as may be inferred from observing the records of drill holes that cut them at varying elevations.

### *Replacement Hypothesis Explains the Great Variety of Phenomena*

The hypothesis of molecule by molecule replacement by ascending, hot solutions and accompanying vaporous emanations explains the great variety of phenomena that have been encountered in the study of iron-ore deposits of Steep Rock Lake. Since this theory so clearly relates all of the important facts now known in connection with the ore bodies, we may have some assurance that, subject to the development of additional facts, it approaches an adequate explanation.

### *Possible Vertical Range of Ore*

On the basis of this explanation of the origin of the ore, there is likely to be a greater vertical range of hematite of the same composition than with similar ore originating from the weathering of sulphides, carbonates or sedimentary iron formations. The extent of the ore bodies, and uniformity of grade of the ore, are indicative of widespread and intensive operation of the same processes throughout.

## ROLE OF HYDROTHERMAL REPLACEMENT IN DEEP-SEATED IRON-ORE DEPOSITS OF THE LAKE SUPERIOR REGION IN GENERAL*

A consideration of the prevailing hypotheses as to the origin of the iron ores is a matter not only of intense interest but of the utmost practical concern. Possibly the only justification for presenting a study of any one area is that, *mutatis mutandis*, the general principles involved shall have some wider application. Accordingly, we shall discuss very briefly hydrothermal replacement processes as they may apply to some other areas in the Lake Superior Region.

Comparing the Steep Rock deposits with producing mines that yield the same type of ores on the Vermilion Range and in the deep mines of the old ranges in Michigan, we find that, despite numerous very obvious disparities, there are many similarities in the mineralogy of the ores, and much the same variety of intrusives in the respective areas, also like deformation and geologic successions. So close are these resemblances that we believe similar replacement processes were active in the genesis of many of the deposits of the Vermilion Range and in some of the old ranges.

Discussing the ore bodies of the Vermilion Range in 1926, Wolff, Muller and Gruner[14] very definitely suggested the replacement of quartz by secondary hematite, calcite and dolomite through the agency of ascending hot solutions. But it was not made clear in that article whether the solutions were thought of as having been the means of conveying iron from deeper magmatic sources or were regarded as having been instrumental in removing silica by hydrothermal leaching and at the same time causing a rearrangement of the iron minerals already present in the iron formations. In subsequent papers, Gruner has laid emphasis on the leaching action of hydrothermal ascending waters rather than on the introduction of iron by metasomatism.

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* An article on the part played by Hydrothermal Replacement in Deep-seated Iron-ore Deposits of the Lake Superior Region has been contributed by the authors to *Economic Geology* (Jan. 1943). In this article the various districts are dealt with in more detail.

### *Difficulties Involved in Theory of Compaction of Leached Ores*

The great difficulty with any theory of leaching, either meteoric or hydrothermal, as applied to the deep ore bodies on the Vermilion or in northern Michigan, extending as they do under thick coverings of jasper along flat rakes to vertical depths of 2000 to 5000 ft., *lies in the very presence of clean, massive hematite in great volume at those depths.* To produce this high-grade hematite it is necessary to add at least 100 per cent of iron mineral to the amount of iron mineral that already was present in any given volume of iron formation in which the ore was developed. On the Vermilion, at least, there is no evidence of the great slumping or compaction that would have been necessary to bring about this introduction of iron mineral from above by mechanical means. Nor can the required compaction be attributed to folding, since the ore bodies were formed within folded structures and subsequent to the folding. This is clear, since the vugs at the periphery of the ore bodies and throughout are fully preserved, with their fragile crystal linings intact. Transfer downward of this great quantity of iron by solution in the zone of weathering from ferric sources within disintegrated jaspers, lying above, does not seem tenable. But there is a reasonable hypothesis that goes far to explain many otherwise unrelated facts; i.e., the supposition that iron from deep-seated magmatic sources has been introduced on a large scale, taking form as hematite replacements, *within brecciated iron carbonates or brecciated, oxidized iron formations.*

### *Sequence of Iron Accessions in Specific Areas*

Considering the whole process over several geologic periods, the deposits may well have been formed by emanations from those same magmatic reservoirs that supplied not only the iron that went into the seas when the banded iron formations were being laid down, but also, after the consolidation, emergence and folding of the iron formations, supplied the later basic intrusives so commonly associated with the ores of the Vermilion, Marquette and Gogebic Ranges. In a final stage of these diastrophic movements, accession of iron by means of heated solutions and accompanying pneumatolytic processes may well have formed large-scale, deep-seated replacement-hematite deposits.

These replacements would also have entailed the removal of the mineral substances so replaced. The essence of the matter is that the two effects were probably simultaneous. The solutions, having dropped their burden of iron in the replacement of banded iron carbonates, ferruginous chert, jasper or greenstone, may have made their way on upward, carrying a load of silica instead of iron. No doubt the upper portions of the deposits were reworked later by surface waters, masking earlier, more deep-seated effects. On the other hand, the development of ore occurring in the relatively shallow deposits of the Mesabi Range was doubtless largely the result of the leaching of iron formations and subsequent compaction by slumping.

### CONCLUSION

By way of summary, from the standpoint of the explorer, it is necessary to realize that many different geologic processes may operate in the transfer of iron minerals. The difficulty is to distinguish between the effects of these processes in any particular instance and to be guided accordingly.

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## DISCUSSION

Stephen Royce,* Crystal Falls, Mich.—Having examined and reported upon the Steep Rock Lake property and vicinity on several occasions for clients, and having kept in touch with the operation from 1938 on, the writer offers the following discussion of the valuable paper by Messrs. Roberts and Bartley, which was so kindly prepared in response to the writer's earnest request, made as a member of the Committee on Mining Geology.

### Origin of the Ores

The writer wishes to express complete agreement with the authors that the origin of these deposits was replacement action by hot ascending sulphide solutions of igneous origin, gaining access via the limestone and dolomite breccia channels, and acting mainly to replace the dolomite and limestone breccia with hematite and hydrated hematite.

Some replacement in adjoining greenstones and tuffs by the same solutions may also have taken place, but the latter occurrences may represent replaced iron formations originally bedded in with the volcanics. Iron formation in the same relationship to volcanics is extensively developed on the western Menominee Range of Michigan, in the Middle Huronian belts; also appears in similar relations on the Vermilion Range; and is abundantly exposed in the same stratigraphic position no farther away than the southeast arm of Steep Rock Lake itself.

Furthermore, the writer's classification of drill cores, made June 11, 1938, on the property, contains the following, for hole 8, in the original strike of the *A* ore body: "332–342 ore; 342–350 much-leached sugary chert with thin ferruginous slate bands."

This is a typical iron formation. The ore in this first group of holes (Numbers 6 to 13, incl.) is described in the writer's 1938 notes as follows: "The ore is hard, massive hematite with some limonite, and shot through with many small vugs." The breccia forms in the float ore and in the drill samples were also noted. As a field classification this does not check badly with Professor Schwartz's microscopic work.

Notes supplied by Mr. Neeland, then geologist at the exploration, show: "Hole 23 has some iron formation, not logged yet." It appears therefore that there is some iron formation at the exploration, much iron formation in similar stratigraphic position in the vicinity, and iron formation similarly located as to volcanics on the Vermilion and western Menominee Ranges. Certain of the holes, notably Holes 25 and 27, showed some specular hematites resembling those known to be formed from cherty iron formation elsewhere.

Hence the writer concludes that at least a small part of the Steep Rock ores results from replacement of iron formation. By far the greater part of it is, as the authors say, clearly a replaced limestone and dolomite breccia, with the breccia outlines still preserved in much of the ore, and all of the replacement is by hot ascending waters.

The authors' citation of the neighboring Atikokan pyrite, pyrrhotite and magnetite deposits in greenstone, as indicating the type of solutions acting in the vicinity, is most illuminating. Such solutions clearly carried $FeS_2$ and FeS in solution, no doubt in superheated steam. The limestone and dolomite breccias, best developed at the fold corners of the southward-pitching capital M into which Steep Rock Lake and its surrounding rocks are bent, are virtually co-extensive with the ore bodies, both horizontally and vertically. They are loose, cavernous, rubbly, broken rocks, cemented only by the ore bodies along their southward and upper edges against the overlying tuffs and greenstones. They are ideal channels for solutions that moved upward in them, dammed on the south by the less permeable volcanics. Rising hot solutions would circulate most actively against these steep roof surfaces at the top of the breccia, and there the chemical replacement action would be strongest right where we find the ores. In lesser quantity the ore minerals are widely scattered in the breccia.

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* Mining Engineer and Geologist.

In the presence of superheated steam at high temperature, carrying FeS and $FeS_2$ in solution, we get:

$$CaCO_3 + H_2O = CaCO_2H_2 + CO_2$$
$$3CaO_2H_2 + FeS + FeS_2 = 3CaS + Fe_2O_3 + 3H_2O$$

Excess of water would hydrate much of the $Fe_2O_3$, as we find it. The process would continue until all unsealed voids were filled, leaving only small, separate vugs unfilled. The steam, carbon dioxide, and soluble calcium sulphide would go out up the breccia to the contemporaneous surface, now long since eroded, where it would deposit calcite and free sulphur, both widely known in hot-spring deposits. The above reactions are offered, not as the last word, but as an illustration of the action to be expected from pyrite-pyrrhotite solutions in this setting.

## SIZE OF ORE BODIES

The tonnage figures given are very conservative as compared with total tonnage probabilities. The deep overburden of glacial clay, taken in conjunction with the relatively narrow width and steep dip of the ore bodies, from an open-pit standpoint, makes it clear that Steep Rock will be, in the main, an underground operation, with open pitting restricted to its earlier years only. But it seems obvious that a group of ore bodies having a proved ledge outcrop area of something over a million and a quarter square feet; a proved depth below ledge of 1200 ft. at one point and 1000 ft. at another; a clear-cut relationship to a steeply pitching fold structure and associated breccia channels which cannot be expected to cut off at any shallow depth; a definitely deep-seated rather than superficial origin; a cubic factor of not over 10 cu. ft. per ton; it seems obvious, to repeat, that such a group of ore bodies can hardly be measured by the twenty-five million or so tons which the authors have put into figures. This comment is made, not in criticism, but to point out the extreme caution with which their estimates have been made.

## CORRELATION

This view of the Steep Rock picture is to the writer almost the most significant feature of the whole interesting show. Essentially this writer is in agreement with the series as set up by the authors, which confirms in almost every detail the succession reported by this writer in 1938 for clients.

In slightly more detail than the authors have shown, the writer lists the series as it appears at Steep Rock Lake and vicinity, from top to bottom.

### *Post Steep Rock*

A series of southward-dipping clay and graphitic slates with prevailingly east-west strike outcropping along the railway and southward thereof in the vicinity of Atikokan and westward. These slates ride right across the open ends of the M-structure of Steep Rock Lake, unaffected thereby, and are believed to be unconformably younger than the Steep Rock series. These slates are mapped by Dr. Leith[15] and associates as Knife Lake slates. The same authority shows them intruded by granite some miles to the southwest of Atikokan, just as the Knife Lake slates are intruded by a granite on the Vermilion. The Canadian geologists have mapped them as the Seine series or as the Coutchiching series. That they are equivalent to the Knife Lake slates, and unconformably younger than the Steep Rock series, this writer emphatically agrees. On the Vermilion Range the Knife Lake slate rides unaffected over the complex trough structures of the iron formation and greenstone, on the eroded raw edges of which plasters of the basal Knife Lake conglomerate (Ogishke) are found in profoundly unconformable relations. Similar isolated remnants of conglomerate are shown on the eroded surface of steep-dipping folded volcanics south of Steep Rock Lake, as mapped by E. S. Moore,[16] which this writer believes represent plasters of the basal Knife Lake or Seine series conglomerate. The granite intrusive into these Seine or Knife Lake slates is of course also post Steep Rock, and would also have an intrusive contact with that series, as exemplified in the granite north and west of Conglomerate Bay, where the Steep Rock *A* ore body lies. This latter granite carries inclusions of limestone and dolomite, of an earlier granite, of greenstone, and of iron formation. It is probably the same granite that intrudes the higher Seine slates, but is nowhere traceable to it on surface.

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[15] G. K. Leith et al.: U. S. Geol. Survey *Prof. Paper* 184.

### *Erosional Unconformity*

*Steep Rock Volcanic and Iron-bearing Series.* The U. S. Geol. Survey *Monograph* 52 and *Professional Paper* 184 correlate this series with the Ely greenstone of the Vermilion Range. With this relationship the present writer is in complete agreement.

The series consists of a thick succession of greenstone flows, tuffs, and volcanic agglomerates, with steep southerly dip, the flow tops being clearly to the south. It grades downward into ash or tuff beds, with thin intercalated flows, which at their bottom grade downward into an iron formation on the southeast arm of the lake, and into the ore bodies at the Steep Rock exploration.

At the bottom of the ore bodies, or at the top of the underlying rocks, drilling has shown a thin, discontinuous quartzite selvage, like the Traders' quartzite of the eastern Menominee Range. Below this is a talcose schist or slate (Hole 25), below which is a conglomerate containing boulders of granite, greenstone, dolomite and limestone. This exactly duplicates the eastern Menominee relationship and shows an erosional unconformity below the iron-bearing series and beneath the volcanics.

*Steep Rock Limestone-dolomite-graywacke-quartzite Series.*—This consists of a series of limestones and dolomites, which in the neighborhood of the ore bodies are intensely brecciated into a loose rubble, cemented only enough for a loose cohesion, but in no degree sealed. The percentage of voids is great, and the size of the voids from minute to cavernous. Along the southeast arm of the lake the series is unbrecciated and well exposed. The dolomite limestones there show algeal (or possibly concretionary) structures like those of the Kona and Randville dolomites on the Marquette and Menominee Ranges. These structures, convex to the south, show the top of beds to be southward, as is also evidenced by the cross bedding in the quartzites below.

The calcareous sediments grade downward into graywackes and impure quartzites like the Mesnard and Sturgeon quartzites of the ranges cited. These in turn rest on a gray Laurentian-type granite, separated from it by a conspicuous sedimentary conglomerate containing granite and greenstone pebbles and boulders, as well as quartz pebbles. Areas of greenstone of Keewatin type are to be found in the granite in the neighborhood.

*Laurentian-type Granite with Areas of Keewatin-type Greenstone.*—This entire series duplicates the Vermilion Range down to the iron formation; it duplicates the Menominee from top to bottom; and it duplicates the Marquette with the exception of the greenstone, which feathers out between the Menominee and Marquette, and is also missing on part of the eastern Menominee. It also duplicates the Gogebic, below the Keweenawan, if account be taken of the conformable Middle Huronian greenstones east of Wakefield.

The granite relationships duplicate those of the Vermilion, with a Laurentian, and a post Knife Lake granite; of the Gogebic, with Laurentian and post-Tyler granites; of the Marquette, with Laurentian and post-Michigamme granites; of the Menominee, with Laurentian and post-Michigamme granites.

The entire series of the Atikokan area is also duplicated in interior Labrador, except that the writer saw no positive occurrence of post-upper-slate granite there, though late intrusive granite is found in the basement of the series. The entire series from the greenstone up is developed in the pre-Cambrian at and near Jerome, Ariz., including a specular hematite jasper within and at the top of the greenstone; diorite intrusives; overlying folded schistose slates (Yavapai schists); and a late pre-Cambrian intrusive granite near Prescott. The whole Menominee-Marquette series is duplicated in India, in the same order, by the testimony of several Indian Geological Survey men who have visited the Lake Superior region with the writer. Specimens corresponding to the Lower, Middle, and Upper Huronian of Michigan were collected by Dr. C. K. Leith in India, which matched, formation by formation, with corresponding specimens taken on the Marquette and Menominee Ranges by the writer while Dr. Leith was collecting the Indian specimens. Dr. Leith mentioned these correspondences in his presidential address to the Geological Society of America.

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[16] E. S. Moore: 48th Ann. Rept. Ont. Dept. Mines (1939) **48**, pt. II.

What these amazing correspondences mean, at such enormous distances, is of course open to argument. In talking with the writer about these widespread repetitions, Dr. A. C. Lawson expressed approximately the same opinion suggested by the writer in an article on the Lake Superior iron ranges, reprinted in the 1936 *Transactions* of the Lake Superior Mining Institute, which was delivered at the American Mining Congress' Chicago meeting Sept. 23–27, 1935, and published in the *Mining Congress Journal* of March 1936. As Lawson expressed it, in 1936, if memory serves, these widespread correspondences probably reflect the fact that continental movements, and cycles of diastrophism, in those early ages of a primeval world, were more or less simultaneously duplicated around the globe. The correspondence in the southern hemisphere has not so far been shown to be as close.

At any rate, with all its complications, the pre-Cambrian increasingly shows certain worldwide correspondences and simplicities, which can be applied practically, with due caution, to the solution of certain local correlation problems. Within and about the North American pre-Cambrian shield, on whose edge lies the Lake Superior region, these correspondences are clearly very close. Labrador, the Michigan and Minnesota Ranges, and the Atikokan area, are all but different facets of the same stone.

Applying the correspondence noted, the Steep Rock area correlates as follows, in the writer's opinion:

Seine-Knife Lake slates and intruding granites, Upper Huronian.

Steep Rock-Ely greenstone-iron formation series, Middle Huronian.

Dolomite-quartzite series, Lower Huronian.

Basement granite, Laurentian.

Certain basement greenstones within the granite, Keewatin.

Ely greenstone is classified as Keewatin by most authorities. Keewatin usually is said to be intruded by Laurentian. (The writer has never seen the relation of indubitable Keewatin to indubitable Laurentian.) Yet the Steep Rock greenstone, called Keewatin, is above, and separated by two unconformities from, a basement granite, which is called either Laurentian or Algoman by various authorities. How can it then be Keewatin?

The writer realizes that the proposed correlation will meet much opposition, but it brings the Michigan, Minnesota and Atikokan sections into complete agreement with one another and with the Labrador section, an agreement conspicuous by its absence heretofore. It throws the Michipicoten into the Middle Huronian, as well as the Vermilion.

It also throws a big slate series of the Sudbury-Cobalt area from Lower Huronian to Upper Huronian, corresponding to Virginia, Michigamme, Tyler, and Seine-Knife Lake. All these series are of great thickness, contain graphitic slates, graywackes, and a high-phosphorus iron formation. Again, this correlation agrees fully with a series examined by the writer east of the Sault and north of Bruce mines in 1940, where dolomite and quartzite of Lower Huronian type were found unconformably below a graphitic slate series with remnants of Middle Huronian specular and red jasper and hard ore in the conglomerate at the unconformity, as well as a few Ajibic quartzite remnants beneath the upper conglomerate.

### Igneous Action in Ore Concentration Elsewhere in the Lake Superior Region

The writer finds himself fundamentally at variance with the authors regarding igneous action in ore concentration elsewhere in the Lake Superior region, and bespeaks the same freedom of expression for his disagreement that he has already been at some pains to obtain for their expression of their views. Scientific knowledge can advance only as it stands the test of discussions of this sort. New theories are advances of knowledge only if they can meet and overcome objection; and, by the same token, established theories should stand only if they can meet and answer new ideas and theories. The last word in ore deposition processes will never be written while the race continues to seek and produce minerals, but ideas are not necessarily advancements just because they are new. These things must be thrashed out, and technical and scientific literature is the place where this must be done if the fraternity in general is to benefit. From such an open forum, only the truth can finally emerge. Such a forum was supplied by this Institute in the formative period of ore-deposition theory, in famous volumes of the Transactions now almost priceless, where the theories of D'Aubrée, Posepny, Vogt, Sandberger, LeConte, and many others, were developed. It is from this discussion that the modern science of ore deposition has evolved. The Institute is vitally concerned in its further evolution, and should foster it. For it is only the truth about all types of ore deposition that will enable men to find the hidden ores of the future. The ores one stumbles upon are mostly either in present production or are already knit into the metallic skeleton of civilization.

The authors refer only briefly to this phase of their subject in the present article, reserving most of their case for presentation in the January 1943 number of *Economic Geology*. In part the writer cannot avoid referring here to the wider phases of the problem which they there present, but will hope to discuss this question more fully through that other medium.

In the first place, this writer would refer to an old physiological conundrum: "Why does not the stomach digest itself?" By now that question has been pretty fully answered by the physiologists, but as to igneous rocks that supply ore minerals for replacement, and the filling of great cavities dissolved in the host rocks by igneous solutions, the answer is, "They do digest themselves!" At Bingham, Utah, the limestones about the edges of a great porphyry mass have been subjected to molecule-by-molecule replacement with heavy sulphides, which obviously have issued from the porphyry. The channels are there, the leaching and mineralization dies out away from the porphyry, and builds up toward it. Does the porphyry itself show mineralization? The answer to that is the greatest open-pit copper mine in North America, mining the mineralized periphery of the porphyry. In less spectacular fashion, Ely, Nev., Bisbee, Ariz., and other great copper mines of the southwest display similar relationships. At Butte, Mont., the channels of access for igneous solutions require little search and no imagination to disclose them. No trouble is found in tracing the igneous origin of the Comstock lode, and little with the Mother Lode mines of California.

In a few areas, like the Coeur d'Alene lead mines of Idaho, the channels (veins) are clear enough but their continuity to an igneous source is inferred rather than traceable by continuous exposure. In other cases, like the Tri-State zinc-lead district and the Rosiclare fluorspar district, the igneous origin is inferred from the mineral assemblage and the vein channels, but the igneous source rock is hidden.

In no generally recognized case of known igneous origin, where the source rock or the transferring agency is exposed, does one need any imagination to see the igneous connection, and to decipher the approximate character and mode of access of the ore-forming solutions. At Steep Rock all these tests are met. The source rock is clearly the later granite, which is shot through with small gold-bearing quartz and sulphide veins. In the neighboring greenstone are pyrite-pyrrhotite deposits of vein type and clearly later igneous origin. Such solutions, active in the neighborhood, would find ideal channels in the calcareous breccias, and would naturally react with them to form the hematite and goethite that are so abundantly found.

Now let us apply some of these same tests to the Lake Superior iron ranges.

### *Gogebic Range*

Where was the source rock from which the hot, upward-moving, hematite-bearing solutions are claimed to have reached the Gogebic Range iron formation? Was it the numerous dikes, which cut across the formation, and upon which the ore bodies are found in troughs formed between the dikes and various impervious beds within and at the bottom of the iron formation? These dikes are leached, weathered, and oxidized in contact with the ore, as is the iron formation. But in the footwall, and in the granite beneath, through which they came, these same dikes are hard, fresh, black, massive diabases and diorites, wholly unaffected by solutions or channels, wholly without porosity, and without any sort of effect on their wall rocks. They look like typical crystallized dry melts. They have not digested themselves or their surrounding rocks in the minutest degree. They are just the same where found in the hanging slates and in the overlying Keweenawan traps. Again they are just the same where encountered in unoxidized iron formation below the depths to which oxidation has penetrated. Beyond the sparsely scattered pyrite and magnetite crystals, universally found in such rocks, these dikes have no sulphide or iron oxide minerals whatever. Their iron content is in the form of silicates.

The total lack of leaching of their wall rocks except where in contact with ore and leached iron formation shuts off all chance that the dikes have filled pre-existing channels in which hot mineralizing solutions traveled.

Did the later granite bring in the mineralizers? Where this is best developed, on the east and west ends of the range, there are a number of quartz veins cutting the iron formation, and wherever these are present in quantity there is no ore. Occasionally such veins are found cutting ore bodies, but they are simply "bull quartz" veins, with a very few pyrite crystals, and very rare specular hematite crystals, and have no observable effect on the surrounding ore and iron formation, beyond silicification for a few inches in their walls.

Was the source rock the later intrusive Keweenawan gabbro above the iron-bearing series? As this attains size and proximity to the formation toward the west, the formation becomes an amphibole magnetite schist, with only such little specular hematite as can easily have been later indurated from the results of the leaching effect of pre-Tyler slate weathering.

Did the solutions come up the formation from depth, possibly along the great bedding fault, and the cross faults? If so, then why did the ore not form *under* the obstructing dikes as it forms *under* the impermeable volcanics at Steep Rock? The bedding fault displaces the dikes in the unoxidized carbonate iron formation beneath the western Ironwood mines just as much as it does above. But no solutions have come up here to alter the iron formation. Why not?

The compaction of residual hematite bodies at depth without addition of hematite requires, as the authors say, the creation of great voids and the slump of overlying formations into them. This is just what we find on the Gogebic, most conspicuously developed. Our largest ore bodies there have always been accompanied by much evidence of slumping in the hanging wall and of natural stoping up the dip. The tops of the ore bodies are commonly accompanied by loose, jagged rubbles of caved iron formation, by great vugs, and by open natural stopes. By these phenomena we have learned to recognize the existence of ores beneath us. As the Cambrian basal conglomerate abounds in ore specimens, it appears that the ore bodies were about as they are today in basal Cambrian time. Therefore these must be some of the world's oldest rock openings.

The Gogebic ores are found in upward-opening trough structures, always communicating through ore or well-oxidized iron formation, and/or through fault channels with ledge surface. Upward-moving hot waters would deposit their minerals *under* the obstructing dikes, but the ore bodies are *on top* of them.

And now let us leave the question of where the alleged hot waters came from and ask, what did they carry? Certainly not sulphides, for the whole area is unusually free from such minerals. There are no near-by pyrite-pyrrhotite deposits, as at Atikokan, and no gold-quartz-pyrite veins in the vicinity. There is no pyrite in the neighborhood beyond the usual igneous minor minerals and the usual original pyrite content of the carbonaceous slates that overlie the formation. Anyone who has ever smelt tidal mud flats knows where *that* sulphur came from.

Did the hot solutions carry hematite only in solution? Hematite is one of the least soluble minerals. The rare botryoidal hematite vug linings show that minute quantities of hematite have traveled in some kind of solution. The few tiny specular hematite crystals that form mineral curiosities in the rare occasional quartz veins show that *some* hematite once traveled in hot water in the area. But evidence that hematite ever moved in hot solution by the millions of tons in this area is wholly lacking, and the writer knows of no such evidence in the entire world.

### *Marquette and Menominee Ranges*

Space forbids detailed analysis of these, which is reserved for a later discussion to be submitted to *Economic Geology*. The same general arguments apply there also. The relations of the ores there are mainly to the bottoms of upward-opening structural troughs. These are fold troughs, fault troughs, and intersections of dikes with such structures. Fault intersections with the iron formations form solution channels whose effect on ore concentration corresponds closely with that of the cross faults on the Gogebic. Fold breccias, especially in the upper horizons of thick iron-bearing formations, have an important effect on oxidation and ore concentration, which always terminates downward and increases upward, suggesting downward-moving solutions.

Typical ore settings on all the ranges are diagrammed in the writer's article published as Chapter II of "Lake Superior Iron Ores" by the Lake Superior Iron Ore Association of Cleveland, Ohio, in 1938, which will serve also to illustrate the earlier article by this writer cited above, as well as the present discussion. The same diagrams, and a discussion of Lake Superior iron-ore formation, appear in the Symposium "Ore Deposits as Related to Structural Features," published by the National Research Council in November 1942, as the writer's contribution thereto.

### *Vermilion Range*

Reference is here made to the block diagram of the Ely trough appearing in the writer's chapter in "Lake Superior Iron Ores" cited above, or to its duplicate published in the symposium mentioned. The scale and purpose of the diagram, and the confidential nature of its basic information, precluded the display of more than the major structural features in a generalized diagrammatic fashion.

Mr. Roberts and Mr. Bartley find difficulty in accounting for the necessary slump and compaction required by concentration of hematite by removal of silica and residual enrichment of the hematite. The ores at Ely were, as stated by the authors, clearly formed in a preexisting trough. The narrow width of the jasper renders difficult any slump wholly within the formation sufficient to close the voids that would necessarily result from such a large concentration of ore. What they overlook is the overlying greenstone hanging, also caught into the trough. This would slump, and undoubtedly has slumped down as a body into the voids created by ore concentration, just as it is today subsiding into the approximately equal voids formed by mining the ore. That the present subsidence scarps are following old movement cracks is clearly shown at a number of points.

The authors refer to the uniform narrow width of the jasper horizon as evidence that sufficient slump cannot have taken place to allow of residual concentration. The Zenith mine is on the south limb of the trough, in the middle block of the block diagram above referred to. Its ore body pinches out upward along the footwall of the iron formation, not reaching ledge by several hundred feet. The detail cross sections of the mine show the ledge outcrop of the unconcentrated iron formation reaching widths of over 300 ft., and never less than 200 ft., while the footwall and hanging-wall greenstones rarely exceed and usually do not reach 100 ft. apart normal to the bedding in the lower reaches of the ore body. Here is just the most perfect evidence of the slump they find wanting, and whose alleged absence leads them to postulate introduction of hematite by hot solutions, which must have come from an unknown source, via channels not demonstrated to exist, carrying materials of unknown composition.

To sum up the igneous setting of the Vermilion iron formation: It was laid down in the Ely greenstone epoch. It is interleaved with thin, contemporaneous flows, whose vesicular tops preclude their interpretation as dikes, and assist materially in determining tops and bottoms of beds in folded and overturned areas. A very few diorite or diabase dikes cut across the formation, but show no ore-forming influence differing from that of the Gogebic dike troughs.

A later granite intrudes across the unconformably younger Knife Lake slates. Where this intrudes the iron formation we find the black magnetic, or "hungry" jaspers too well known to all Vermilion explorers, and described by Clements.[17]

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[17] Clements: U. S. Geol. Survey *Monograph* 45.

We thus see that the vugs (incidentally they are rather rare at Ely) referred to by the authors become, not an argument in favor of igneous solution origin, but a remnant of openings produced by leaching and not entirely closed by slump. Incidentally, at Ely, the writer knows of no small, discrete, noninter-communicating vesicular vugs in the ore, like those of Steep Rock. Few Ely openings seen by the writer are lined with crystals of any mineral.

The Ely ores are prevailingly specular hematite, which Clements describes as always hard. The so-called "soft ores" he finds to be due (p. 183 ff. supra) to brecciation of the hard ores, later explained as due to some further moderate plication of the folds after the induration of the hard ore. He then describes a later infiltration of soft red ore into the interstices of this breccia. At the Zenith we produce a high-grade hard ore by simply screening out this later soft infiltration ore and shipping the breccia pieces as hard ore. The later brecciation is absent at Soudan, where the hardest ores are found.

Throughout the Lake Superior ranges hard ore and specular hematite in iron formation are always accompanied by a history involving weathering action in an earlier erosion cycle followed by specularization and induration in a subsequent period of diastrophism. (Marquette hard ores and jaspillites; Menominee blue cellular hematites, oxidized in pre-Michigamme time and specularized in post-Michigamme time; certain similar blue porous ores on the Gogebic.) Where no post-ore diastrophism has taken place, red, earthy hematites and hydrated hematites are found, as on the Mesabi and Cuyuna Ranges and in the Upper Huronian of Iron County, Michigan, and of the western Marquette. Certain specular hematites on the western Mesabi are found where upper iron-bearing members have been removed by pre-Virginia slate erosion, and seem to reflect pre-Virginia weathering followed by post-Virginia specularization.

On the Vermilion the major folding preceded the Knife Lake, because the latter slate, and its basal conglomerate, are found across the upturned eroded edges of the iron-formation folds. The ore must have been formed after the major folding because it lies in these folds in typical Lake Superior fold-trough fashion. The Knife Lake slates have been later tilted and folded, and in this period the hematites must have been indurated and specularized. In certain areas minor further plication of the original major folds in the iron formation brecciated the hard ores in varying degree. Subsequent weathering in the present or the immediately pre-Cambrian erosion cycle, or both, has produced the minor amounts of soft, red, earthy hematites.

In closing this phase of the discussion, reference to the history of ore-deposition theory shows that the worst of the early errors were not so much in the theories as in their over-wide application. The early investigators argued over whether ore bodies in general were caused by hot, rising igneous waters; by cold descending waters; by subsequent enrichment of lean protores, and whether this was by hot rising or cold descending waters; whether they were originally laid down as they now exist; whether they were magmatic segregations or sedimentary beds; whether they were formed by horizontal leaching, and redeposition called lateral secretion; and so on almost indefinitely. We know now that *all* of these modes of formation applied to some ore bodies and not to others, and that the major errors arose from trying to apply in the wrong place theories that were right in many other instances. Even the almost discredited lateral secretion theory still explains most of the cold-water milky quartz veins better than any other.

When the senior author first explained his ingenious application of the pyrrhotite-pyrite hot solutions to the Steep Rock problem to this writer, who fully concurred, he said he thought the very factors that prove hot water at Steep Rock disprove it on the Lake Superior ranges in general. With this view the writer still concurs, and feels that the authors, in their natural enthusiasm over so important and novel a situation, which they have so beautifully unraveled and portrayed, have erred in trying to spread this mode of origin over areas where its basic factors are wanting, and where the recognized theories fully accord with all the facts yet known.

### FEASIBILITY OF THE STEEP ROCK PROJECT

The authors express the opinion that there is no geological factor so adverse as to preclude the mining of the Steep Rock ore bodies. With this general statement this writer agrees. The water factor, of course, is the great obstacle. No major iron-ore body outcropping under a deep lake can be mined without removing the lake. This was recognized at a rather late date by those who were then in charge of this enterprise.

The removal of the lake entails not only the diversion of the Seine River and the pumping of the lake, but the additional pumping of the entire rainfall on the watershed of the drained portion of the lake, less whatever evaporation takes place. Climate maps for the area (London *Times*, Hammond, Creager and Justin's Hydroelectric Handbook) agree closely on 27 in. mean annual rainfall. The last authority indicates about 10 in. annual runoff, and a mean evaporation constant of possibly 50 per cent. This leaves for seepage into the subsoil some 3 or 4 in., which will mostly reach the lake area through the various faults that communicate with the footwall breccias. This means that not only during the period of pumping the lake, but permanently thereafter, some 13 or 14 average rainfall inches per year will have to be pumped for the entire watershed of the drained area. This translates into a good many thousand gallons per minute. A further unknown element of hazard is that fault fractures may bring additional water from outside the local watershed.

With virtually unlimited cheap electric power available this flow ought to be possible to meet, and there seems little doubt that the size of the ore bodies will warrant meeting it.

The all-important factor to be figured into all plans, for the entire life of the mine, is the enormous open footwall breccia. This outcrops on the shore and under the waters of the lake. It has been shown by angle holes as well as by surface outcrop to be up to hundreds of feet thick and nearly coextensive with the ore bodies, which by the nature of their origin it could be expected to be. Its enormous volume of voids must hold an amount of water second only to the volume in the lake itself.

Unless this breccia is kept drained, to current mining levels, any opening that strikes it, or strikes any open fault fracture communicating with it, will encounter the full vertical difference of head to wherever water level stands in the breccia. This necessarily follows from the most elementary principles of hydraulics. It is fully in accord with mining experience elsewhere. It was forecast by this writer in 1938 and many times subsequently. And it happened at Steep Rock when the attempt was made to open the ore body underground without draining the lake.

On the Menominee Range the same dolomite footwall contains fault breccias, incomparably smaller than these fold breccias, and outcropping under hills, not under lakes. In the early days of the district these relatively small breccias, filled with water and communicating often with one another and with adjoining mines, caused many floods, some disastrous to human life, and all of serious nature, until measures were taken to keep them drained to working levels. Then they became allies instead of enemies. By simply closing a valve on a drainage hole, water can be backed up in these breccias for hours, days, or even weeks when pumping conditions render it advisable, on account of current outages, peak loads, or necessary plant repairs or alterations.

The water problem at Steep Rock is therefore not confined to simple diversion of the river and removal of the lake. It will, in important measure, always be present. It is a large factor, but it is surmountable at a price.

## CONCLUSION

The Steep Rock ensemble is unique among large iron deposits in its geologic setting. One of its most interesting features is this very individuality of environment and origin. It throws important light on the whole problem of correlation in the pre-Cambrian of Canada and the United States.

We are greatly indebted to the authors, and to their principals, the management of the Steep Rock enterprise, for this early, detailed, and skillful analysis and exposition of so interesting an occurrence, as well as the permission to publish it.

## IGNEOUS ACTIVITY RESPONSIBLE

E. S. MOORE,* Toronto, Ont., Canada.—There is much difference of opinion concerning some of the stratigraphic features of the Steep Rock area, but this is not the place to enter on an extensive discussion of this matter. I wish, however, to remark, in passing, that the lack of any age relation between the rocks of the Steep Rock series and the so-called Seine rocks to the south, mentioned in the paper, is still an open question. Further, I consider that there are granites of two ages in the area.

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I am fully in accord with the authors that there is no typical *banded iron formation* in the Steep Rock series, although there is some with the Keewatin greenstone in the area. Other geologists have claimed that the banded formation is present in the Steep Rock series and they have attempted therefore to compare the deposits with certain others of the banded type. On the other hand, I see little of the similarity mentioned between the Steep Rock and the Nastapoka Islands deposits in Hudson Bay. The latter are in the same series as those on the Belcher Islands and certain other islands in Hudson Bay, with which I am familiar, and which are of the sedimentary, banded jasper type. Further, I do not see much justification for linking the Steep Rock deposits with those in the Atikokan Range, about 20 miles to the east. They are rather high-sulphide, mainly magnetite deposits, in rocks quite different from those of the Steep Rock series.

When the work for my map of and report on the Atikokan area, to which reference is made in the paper, was done the iron deposits under Steep Rock Lake had not been discovered; only float ore; up to that time, having been found. A large area was covered during the summer with the able help of a number of assistants. A great variety of intrusive igneous rocks, regarded as belonging to the Algoman sequence, were found outside of the Steep Rock Lake section of the area. At that time the evidence of a genetic relation between the Algoman igneous activity and the primary ferriferous minerals in the Steep Rock series, however, did not appear to be definite. When examining some properties west of Steep Rock Lake, three years later, I found what was regarded as definite evidence of carbonization of the Keewatin rocks by Algoman igneous action on a considerable scale. Such carbonization is a very characteristic feature of many other areas where Algoman intrusives occur in the Canadian shield.

The opinion that I have held from the beginning is that igneous activity (now considered Algoman) was responsible for replacement of rocks of the Steep Rock series—limestone, lavas and tuffs, and some Keewatin greenstone, by siderite, ankerite, dolomite, and some sulphides. The ferriferous minerals were then oxidized to an unusual depth by meteoric waters, in zones highly disturbed by faulting and brecciation. The porous and cavernous ore and limestone, and the nature of the ore minerals, as described by the authors and by Smith,[18] all point to a situation under which supergene iron deposits have been developed.

The conditions recall those seen at Bilbao, Spain, where large quantities of hematite and hydrous iron oxides have been developed by the oxidation of siderite, which through igneous action had replaced limestone. The engineers at the Bilbao mines had a saying, "No faults, no ore." There was a shallow mantle of oxides over the siderite, but these extended down hundreds of feet along faults. The oxide deposit, now worked out at the old Helen mine, Michipicoten district, Ontario is another example of a supergene deposit formed by the oxidation of siderite and pyrite. It extended to at least 800 ft. in depth, although faulting and brecciation such as occurs under Steep Rock Lake were absent, and the conditions were thus in some respects not nearly as favorable for the development of a supergene deposit. There is much evidence that the surface of the Canadian shield in former periods stood much higher with respect to the ground-water table than it does today, and deep oxidation could have occurred under Steep Rock Lake during one of these periods.

## CONSIDERATION OF FAULTING AND MIGRATION OF CaO

E. L. BRUCE,* Kingston, Ont., Canada.—This paper, giving data considered by the authors to favor a hydrothermal origin for the hematite deposits at Steep Rock Lake, will be received with much interest by Canadian geologists. Similar conclusions have been reached by Brown and Morrison[19] in their study of the hematite body at the Josephine mine, in the Michipicoten area, northeast of Lake Superior.

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[18] F. G. Smith: Notes on the Iron Ores of Steep Rock Lake, Ontario. University of Toronto Studies, Geol. Ser. No. 47 (1942) 71.

* Professor of Geology, Queen's University.

[19] E. L. Brown and W. F. Morrison: Geology of the Josephine Mine—Hydrothermal Origin of the Hematite. *Can. Min. Jnl.* (1942) **63**, 5–14.

For many years the apparent lack of major deposits of iron ores in the pre-Cambrian rocks on the Canadian side of Lake Superior has been puzzling and disconcerting. The generally accepted hypothesis that iron ores of the Lake Superior type are related to iron formations naturally focused the attention of prospectors on areas in which iron formations are large, but, in spite of the abundance of them in the Keewatin rocks of Canada, and the large size of some, the search has met with little success.

Collins,[20] suggested that the iron carbonates and pyrite, occurring as lenses with the banded silica-iron oxide iron formations of the Michipicoten area are formed by metasomatic processes. But he considered the lenses of carbonate and pyrite to be integral parts of the iron formation, formed at the same time as the oxide bands, and as part of the same process. Hence his conclusions affected the general conception of the origin of the pre-Cambrian iron ores but little and the guidance of prospecting for them not at all. A metasomatic origin for parts of the iron formations might be accepted without much dissent but few of those concerned with the search for iron ores were ready to accept such a theory of origin for them.

Certain features of the replacements described by Collins indicate rather clearly that they were not formed contemporaneously with the banded iron oxide-silica layers, but much later, after they had been tilted into steeply inclined or vertical positions. Thus the pyrite and carbonate lenses have no genetic relation to the true iron formation.[21] Hawley[22] considers that some of the chert was formed by replacement.

In prospecting for sulphide ore bodies during the last few years, more and more attention has been given to the finding of structural conditions favorable for the emplacement of the ores and less and less to search for a particular rock to which they might be considered to be genetically related. The authors of this paper place many hematite deposits in the same category, so far as origin is concerned, as the sulphide ores. Hence, if their conclusions are generally accepted, the same change of emphasis may be expected to occur in the search for deposits of iron ores.

It is, perhaps, unfair to raise questions concerning the details of the geological relations of the Steep Rock deposits at this early stage in their development, but there are two points in the paper, about which the evidence presented does not seem entirely conclusive. Is it possible that the relations between the Steep Rock volcanic formation and the Steep Rock sedimentary formation are due to faulting and not to unconformity? A zone of fault breccia would explain the conditions described, and such a zone would provide a locus of deposition for the hematite much more favorably than a plane of unconformity.

The paper explains the formation of hematite by the presence of abundant $CO_2$, formed by dissociation of the limestone in the breccia. Hematite bodies, northeast of Lake Superior, lie in volcanic rocks interbedded with some clastics, but with no limestones.[19]

The large amount of $CO_2$ that would be necessary to oxidize ferrous solutions to ferric as postulated would require the dissociation of large volumes of limestone and the escape from the zone of hematite deposition of the CO formed during the oxidation of the ferrous compounds. It would seem that large amounts of new calcium minerals should be formed as part of that process. No such minerals have been mentioned. The only alternative is to assume that CaO was removed completely from the area of limestone dissociation. In what form would it migrate and what would be its ultimate fate?

## SIGNIFICANCE OF GOETHITE AND AN ALTERNATIVE HYPOTHESIS

J. E. HAWLEY,* Kingston, Ont.—The hydrothermal origin of the Steep Rock Lake iron ores proposed by Roberts and Bartley, if correct, is of far-reaching importance in the further exploration of many areas in the pre-Cambrian shield for similar deposits, since their occurrence should depend on the presence of related intrusive igneous rocks and suitable

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[20] W. H. Collins: The Michipicoten Iron Ranges. Geol. Survey Canada *Mem.* 147.

[21] E. L. Bruce: Geology of the Goudreau-Lochalsh Area. Ont. Dept. of Mines, Ann. Rept. (1940) **49**, pt. 3.

[22] J. E. Hawley: Origin of Some Siderite, Pyrite and Chert Deposits, Michipicoten District, Ontario. *Trans.* Roy. Soc. Can. (1942) Sec. IV, **36**, 79–87.

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* Professor of Mineralogy, Queen's University.

host structures rather than on the occurrence and distribution of some original iron-bearing formation.

Before full acceptance is accorded the hypothesis, two points merit much greater discussion than has been given: (1) the admitted failure to account satisfactorily for the presence of appreciable amounts of goethite in what is considered a high-temperature type of deposit, and (2) an alternative hypothesis, which would necessarily require the concentration of iron oxides and hydroxides by weathering processes, their later burial and folding and final alteration to specular ores by hydrothermal solutions arising from some magmatic source.

The paragenetic study has shown goethite to be earlier and replaced by hematite. Tunell and Posnjak[23] have shown "that goethite is unstable at temperatures not much above 140°C.," and their criticism (p. 898) of Gruner's[24] hypothesis, that it is necessary "to assume that the goethite formed and persisted above its maximum stability temperature . . . during an unknown lapse of geologic time," is equally applicable here. Since goethite is earlier than hematite, and since it can and does form at temperatures even below 140°C., it is clear that the high-temperature *inferred* as existing when replacement by hematite occurred, did not necessarily prevail at the time the goethite was produced.

Alternative hypotheses, though discussed briefly by the writers, are dismissed much too hurriedly, without, in this writer's opinion, the consideration they deserve. That the deposits as now known might have been produced by a combination of early enrichment by weathering of some iron-rich rock (not necessarily cherty iron formation), then burial, folding and still later hydrothermal modification, as far as the reader can tell, has received no attention whatever.

The apparent persistence at depth of high-grade ore clearly shows that concentration could not have been effected from the present erosion surface. But what of the erosion surface between the Steep Rock limestone and the overlying volcanics, between which formations the ore is shown? Is it not conceivable that weathering of some iron-rich formation—possibly a carbonate rock—lying above the limestone may have yielded a blanket deposit of residual iron oxides, the extent of which would depend on both the character of the original formation, subsequent erosion and even later shearing and faulting? Considering the present irregular distribution of limestone, as mapped, as well as the limited amount of exploration, it is perhaps not to be wondered at that unweathered remnants of the suggested protore have not yet been found.

Later folding to a vertical position of such residual ore would explain both its persistence at depth and a more normal occurrence of goethite. That the folding did not entirely recrystallize the goethite and bring about its dehydration is possibly a good, though not a positive, argument against this having taken place, for we should note that neither has complete recrystallization of the Steep Rock Lake limestone occurred, and that in it are still preserved remnants of former algal structures.

Still later, following shearing and intrusions, as the authors indicate, may not hydrothermal solutions have entered this zone between limestone and volcanics, altered a large part of the goethite to hematite, dissolved some to make the vugs noted, and even have transported a considerable amount of iron to redeposit it in fractures as well as in the earlier formed ore zone to increase its density? According to such an hypothesis, ore showing a brecciated texture might well represent brecciated ore rather than a replaced breccia for which little real evidence has been given. Silicification and dolomitization of the rocks bordering the ore, and the local development of pyrite, may still be attributable to the later hydrothermal solutions, but whether or not they also contributed additional iron of magmatic derivation is a question most difficult to answer.

Some discussion of the foregoing alternative by the authors would be welcomed, particularly if there are additional facts that make it appear untenable. Otherwise it would seem well to defer full acceptance of the hydrothermal theory until further exploration of the deposits has revealed their true character.

## IN THE LIGHT OF GRUNER'S THEORY

L. M. SCOFIELD,* Laurium, Mich.—Lake Superior geology is indebted to Messrs. Roberts and Bartley for reopening the discussion of hydrothermal concentration of iron ores. Generations of students and visitors to the Lake Superior districts have been unable to accept the large-scale leaching of silica, the migration of iron in relatively high-temperature forms, the relatively high-temperature accessory minerals, and the hot-water types of wall-rock alteration as the work of cool, unaided meteoric waters. Review of the evidence and revision of the basic theory of concentration have long been indicated, and the factual data on the Steep Rock deposits and the authors' interpretations add greatly to the store of information and hypothesis which must be accumulated before any comprehensive theory can be presented. Though certain of the authors' Steep Rock interpretations call for critical comment, their ideas as to the general applicability of hydrothermal influence in iron-ore concentration can be heartily endorsed.

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[23] Tunell and Posnjak: *Econ. Geol.* (1931) 26, 337–343, 894–898.

[24] Gruner: *Econ. Geol.* (1930) 25, 697–719.

* Mining Geologist.

To review the principal points and findings made by the authors, they report iron ore in commercial quantities, consisting of brecciated goethite, cemented by hematite, along an unconformable contact between a limestone and an overlying greenstone series. They correlate the limestone with the similar horizon in the lower Huronian of the south shore, and the greenstone with the Middle Huronian greenstone of the southern province. Not finding an iron formation along the contact in the iron-ore horizon, they have ascribed the source of the iron in these bodies to magmatic contributions, placing the ores in the oxide zone of hydrothermal deposition (490°C.) and ascribing the oxidation to the effect of the breakdown of the carbonate in the adjacent limestone. The authors further suggest that the Vermilion ores and those of some of the old ranges were produced by similar processes, the source of the iron being partly magmatic. While endorsing the authors' idea of hydrothermal influence in the concentration of these ores, it is difficult to accept the idea of magmatic contribution of hematite in the temperature zones that must have existed in the ore-bearing formations at the time of concentration, as evidenced by the accessory minerals.

In 1937, J. W. Gruner (*Economic Geology*) forwarded a modification of his earlier theory of hydrothermal concentration, embracing the idea of actuation of convection currents in the meteoric ground water by igneous emanations, to effect the concentration of iron ores within the bodies of iron formation present on the Lake Superior ranges. The temperatures of these currents would lie within the low-to-moderate range on the hydrothermal scale. With *appropriate* modifications for depth of intrusive, depth of burial of the iron formation, preconcentration condition of the iron formation, and structural conditions governing ground-water flow, this theory can be made to serve to explain all the commercial ores of the Lake Superior region. Variation of these four factors has been responsible for the wide variety in the occurrences and types of iron ore found in the region, a variety that has contributed largely to the confusion in opinion that exists as to the concentration process.

With a belief in the general validity of Gruner's theory as a background, and without denying the possibility in local instances of magmatic contribution of iron, it would seem that the field data presented by the authors for the Steep Rock deposit could be interpreted in another way, which would place the Steep Rock ores on the same basis as the others in the Lake Superior area, though deposited at the high end of the temperature range for such concentrations.

It is generally believed that the bulk of the ores in the Huronian rocks of the region were concentrated after the close of Upper Huronian time and before the beginning of Cambrian time. The authors have correlated the Steep Rock greenstone series with the Middle Huronian on the basis of similarity to the lithologic sequence in the Amasa area, and have related the age of deposition of the ores to the "main time of development of the large iron formations in the Middle Huronian period," but they consider the ores as later than the folding of the Steep Rock basin. The age of deposition given is improbable. The main Middle Huronian iron formations were deposited before the beginning of the mild diastrophism elsewhere marking the interval between the Middle and Upper Huronian depositional periods, and the first deposition after that interval was of a slaty type; therefore, since the authors assign Steep Rock ore formation to the same geologic period that saw the deposition of the greenstone series, either the ore would have had to have been emplaced while the beds were essentially horizontal (obviously impossible), or post-Middle-Huronian folding in the Steep Rock area must have started earlier and been of greater intensity there than anywhere else. The latter is rather unlikely considering the general conformity of dip of the rocks of the two periods and the scanty record of intrusive activity elsewhere during the interval between them. Had the authors assigned the formation of the ores to the period of intense diastrophism and profound intrusion following the *Upper* Huronian they would have avoided this difficulty.

From the evidence presented, there is no reason why the Steep Rock greenstone could not equally well belong to the Upper Huronian period, which includes basic vulcanism in several districts, as exemplified by the Clarksburg volcanics, the Quinnesec pyroclastics, the middle greenstone of Iron County, and the green schists of the Cuyuna. Such a correlation would involve the complete erosion (in the surface section) of the Middle Huronian series in the Steep Rock area—a removal that is a feature of many of the districts on the American side. If this correlation be assumed, the Steep Rock ore bodies then occur at the unconformity between the Upper and Lower Huronian where the great ore-forming iron formation of the Middle Huronian should lie. Although no iron formation was found in the surface section, where the irregularities of erosional removal of the Middle Huronian elsewhere are considered, it is not improbable that at some distance below, down the steep dip of this unconformity in the Steep Rock basin, there may be found Middle Huronian iron formation as a likely source of iron for the Steep Rock ores. The authors themselves report, along with fragments of other alien rocks, banded chert inclusions in the lowest horizon of the greenstone series, just above the unconformity, and these reflect an iron formation somewhere along the course of the unconformity.

In the other units of the Lake Superior geosyncline, ores invariably are found within the iron formations, with a very narrow zone of impregnation and replacement by iron and manganese oxides in the immediately adjacent rocks. Messrs. Roberts and Bartley have emphasized the absolute addition of iron in the formation of the Steep Rock ores and have reminded us of the often overlooked fact of migration of iron in the iron formations of the other units of the Lake Superior region. The solutions that brought this added migratory iron into the ores of the south shore province and produced replacement by iron oxide on a minor scale outside the ore bodies must have picked up the iron from a relatively local source—obviously the iron of the original iron formations. Otherwise there should be replacement ore bodies of hematite along suitable channels entirely outside of the iron formations.

It is suggested that the addition of iron to the breccia along the Steep Rock ore horizon might be more reasonably explained by conveyance of iron from a buried iron formation downdip on the unconformity, by solutions of the proper temperature and composition to accomplish this conveyance, in the same manner as that in which, it is believed, iron has been added to the ground comprising the ore bodies in the rest of the Lake Superior region.

Steep Rock would then differ from the other districts in that its iron formation—probably also containing ores—is the only one that did not come to a land surface after the folding. It would be the only district in which the outlet channel for emanations and the emanation-actuated convection currents could not lie wholly in iron formation, and it would be the only district showing replacement ores in other rocks. This interpretation would obviate the difficulty of explaining why the oxides are present wholly in the ferric state, with no magnetite; why the high-temperature accessory minerals of the oxide zone are missing; and why the whole suite of calcium-aluminum-silicates, which ought to form in such a wall-rock environment at 490°C., failed to occur.

The authors have explained the sequence of oxides (geothite fragments cemented by hematite) as being due to a superabundance of oxygen in the early stages of the process with a decreasing oxygen supply during cementation of the breccia. It does not seem proper to use the oxygen of the $H_2O$ of goethite as evidence of an increased oxygen supply, as surely the oxygen of the water of hydration did not come from the breakdown of the adjacent carbonates. Rather, this sequence would indicate a rising temperature along the channel of replacement, the goethite of the breccia fragments being the earlier and cooler product of replacement and the hematite of the cement constituting the deposition by later and hotter solutions (following the mineralogical associations of these forms of migratory iron in the other districts). This would nullify the chemical significance of the presence of limestone adjacent to the ore body except as an easily replaced material. The significant factors would then become: the presence of a permeable channel along the unconformity, with easily replaced material in the walls, and a source of iron at a depth below, reasonably shallow enough to be reached by emanation-actuated convection currents in the ground water, all in a folded basin of sediments porous enough to permit ground-water convection currents to be set up to considerable depth.

H. M. Roberts (author's reply).—When Dr. Bartley and I undertook to meet Mr. Royce's request for a paper on the geology of the Steep Rock deposits, we could hardly have hoped to bring out these valuable discussions. Because of the difficulty of the subject, we have no doubt come short of dealing completely with many of the related problems. But the gentlemen who have discussed our paper have been most fair, even when they have differed from us. Finally, we have been rewarded most generously for our effort by the acceptance of many significant phases of our work and wish to make our grateful acknowledgments.

# Titaniferous Magnetite Deposits of the Lake Sanford Area, New York

By Robert C. Stephenson,* Junior Member A.I.M.E.

## Abstract

Large deposits of titaniferous magnetite occur associated with anorthosite and gabbro in the Lake Sanford area, Essex County, New York. The ore, gabbro, and anorthosite show consanguineous relations. Anorthosite grades through gabbroic anorthosite into gabbro but is locally intruded by gabbro. Structural and petrographic evidence suggests that gabbro was differentiated from anorthosite by filter pressing. Field and laboratory study of ore occurrences indicates that an ore residuum concentrated in the gabbro through magmatic segregation by normal crystallization and flowage. Ore occurs as bands which grade into gabbro, and also as large masses which replace anorthosite. Ore in the gabbro is finer grained and richer in ilmenite than ore in anorthosite.

## Introduction

The Lake Sanford area lies in the heart of the Adirondack Mountains, in the southeast portion of the Santanoni Quadrangle (Fig. 1), and there are four ore bodies in the area immediately surrounding Lake Sanford. The area lies 30 miles by highway from the railroad at North Creek.

The deposits were discovered in 1826, and from then until 1858 iron was produced in small quantities. They were idle from 1858 to 1906. Between 1906 and 1913, extensive development work was carried on. These early attempts to produce iron ore were thwarted by transportation difficulties and by the titanium content of the ore, which, ironically, has been the element responsible for the present development. In 1941 the National Lead Co., Titanium Division, developed the Sanford Hill ore body and is now producing ilmenite and magnetite concentrates.[16] The Defense Plant Corporation is extending the railroad to the property from North Creek and is building a sintering plant to treat the magnetite concentrate.

## Investigations of the Deposits*

Early studies of the Lake Sanford titaniferous magnetite deposits were made by Emmons in 1842,[12] by Kemp, in 1897–98[18] and by Newland in 1908.[22] Singewald[26] and Osborne[23] considered these deposits along with numerous other titaniferous magnetite deposits in 1913 and 1928, respectively. None of these reports entailed a detailed study of the ore occurrence in this area. The present study was begun in 1940. Seven months were spent in the field, and considerable time was spent in labora-

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This paper was awarded first prize in the Graduate Division of the A.I.M.E. Student Prize Contest in 1943. It was submitted by the author to the Board of University Studies of The Johns Hopkins University in conformity with the requirements for the degree of Doctor of Philosophy in 1943. Manuscript received at the office of the Institute Feb. 16, 1944. Issued as TP 1789 in Mining Technology, January 1945.

* Union Oil Company of California, Laramie, Wyoming.

[16] References are at the end of the paper.

* J. R. Balsley, Jr.,[4] undertook a study of the Lake Sanford deposits following the completion of this report and after the Sanford Hill ore body of the MacIntyre Development, National Lead Co., was developed. The report by Balsley is particularly valuable for its very thorough discussion of the vanadium content of the ore bodies and the estimation of ore reserves. Balsley's geologic map of the Sanford Hill ore body (Plate 19) was prepared during the first summer of the mining operation (1942) and gives the reader a picture of the geology of the developed ore body which was very poorly exposed prior to mining.

TABLE 1.—*Minerals of the Rocks of the Lake Sanford Area*

| Mineral | Composition | Optical Properties | Grain Size | Grain Shape | Alteration Products | Occurrence |
|---|---|---|---|---|---|---|
| Primary minerals: | | | | | | |
| Plagioclase........................ | $An_{52}$–$An_{64}$ | See Figs. 5 and 6 | 0.05–10 cm. | Euhedral to subhedral | Scapolite calcite | Anorthosite-gabbro series |
| Augite........................ | Calcium-magnesium-rich pyroxene[b] | $\alpha = 1.700$<br>$\gamma = 1.745$<br>Birefringence 0.045<br>Max. extinction c∧z 50°<br>Optical character (−)<br>2V 70° ± 5°<br>Pleochroism very weak | Avg. 2 mm. | Subhedral to anhedral | Hornblende chlorite | Anorthosite-gabbro series, anorthosite reaction zones |
| Hypersthene........................ | { $MgSiO$ 82 %<br>{ $FeSiO_3$ 18 %[c] | $\alpha = 1.680$<br>$\gamma = 1.694$<br>Birefringence 0.014<br>Parallel extinction<br>Optical character (−)<br>2V 85° ± 5°<br>Dispersion r > v very weak<br>Pleochroism<br>*X*, red<br>*Y*, pale brownish yellow<br>*Z*, light green | 1–2 mm. | Anhedral | Hornblende chlorite | Anorthosite-gabbro series |
| Diallage........................ | Intermediate between augite and diopside[d] | $\alpha = 1.657$<br>$\gamma = 1.682$<br>Birefringence 0.025<br>Max. extinction c ∧ z −39°<br>Optical character (+)<br>2V 60° ± 5°<br>Pleochroism very weak | 1–3 mm. | Anhedral to subhedral | Chlorite | Gabbro series |
| Green Hornblende........................ | Magnesium, calcium iron, aluminum silicate[e] | $\alpha = 1.658$<br>$\gamma = 1.672$<br>Birefringence 0.024<br>Max. extinction 23°<br>Optical character (−)<br>2V 70° ± 5°<br>Dispersion r > v<br>Pleochroism<br>*X*, yellow green<br>*Y*, olive green<br>*Z*, dark green<br>Absorption $X < Y < Z$ | 1–5 mm. | Anhedral to subhedral | Chlorite | Anorthosite-gabbro series |
| Basaltic Hornblende....... ........ | Iron, calcium, magnesium aluminum silicate[e] | $\alpha = 1.674$<br>$\gamma = 1.713$<br>Birefringence 0.039<br>Max. extinction c ∧ z 13°<br>Optical character (−)<br>2V 80°<br>Pleochroism<br>*X*, yellow brown<br>*Y*, brown<br>*Z*, red brown<br>Absorption $X < Y < Z$ | 1–3 mm. | Subhedral to anhedral | | Ore-bearing gabbro facies, anorthosite reaction zones |

| | | | | | | |
|---|---|---|---|---|---|---|
| Biotite (lepidomelane) | Iron-rich variety[f] | $\alpha = 1.695$<br>$\beta = 1.643$<br>Biref. = 0.048<br>Max. extinction $c \wedge x$ 2°<br>Optical character (−)<br>2V near 0°<br>Pleochroism<br>$X$, pale brown<br>$Y$, brown<br>$Z$, red brown<br>Absorption $X < Y < Z$ | 1–2 mm. | Subhedral | Chlorite | Gabbro, anorthosite reaction zones |
| Garnet[a] | [g] Grossularite 72 %<br>Andradite 18 %<br>Almandite 10 % | Isotropic<br>$n = 1.762$<br>sp. gr. = 3.62 | 2–3 mm. | Euhedral to anhedral | | Anorthosite-gabbro series |
| Apatite | Fluorapatite[h] | $\epsilon = 1.628$<br>$\omega = 1.631$<br>Birefringence 0.003<br>Parallel extinction<br>Optical character uniaxial (−)<br>Colorless | 1 mm. | Subhedral to anhedral | | Ore-bearing gabbro |
| Spinel | Iron, magnesium spinel[i] | Isotropic<br>$n = 1.778$<br>Emerald green | 1 mm. | Anhedral | | Ore, ore-bearing gabbros |
| Ore minerals | Ilmenite-magnetite | Opaque | 1–3 mm. | Subhedral to anhedral | | Ore, ore-bearing gabbros |
| Gangue sulphides | Pyrite-pyrrhotite | Opaque | 1 mm. | Anhedral | | Anorthosite-gabbro series |
| Secondary minerals: | | | | | | |
| Scapolite (dipyrite) | $Ma_{57}Me_{43}$[j] | $\epsilon = 1.545$<br>$\omega = 1.564$<br>Birefringence 0.019<br>Parallel extinction<br>Optical character uniaxial (−)<br>Colorless | 1 mm. | Anhedral | | Anorthosite-gabbro series |
| Chlorite (prochlorite) | Hydrous magnesium, iron, aluminum silicate[k] | $\beta = 1.604$<br>Birefringence low<br>Optical character (−)<br>2V almost 0°<br>Forms rosettes<br>Color green<br>Pleochroism slight | | Anhedral | | Anorthosite-gabbro series |
| Carbonate (calcite) | $CaCO_3$ | $\epsilon = 1.488$<br>$\omega = 1.658$<br>Birefringence 0.170<br>Optical character uniaxial (−)<br>Colorless<br>Rhombic cleavage | | Anhedral | | Anorthosite-gabbro series |

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[a] The index was determined to be 1.762 for a number of garnet grains but only one specimen was sufficiently large and free from contamination to permit a gravity determination.

[b] Larsen and Berman, ref. 19, p. 244.

[c] Winchell, ref. 31, p. 177.

[d] Ford, ref. 14, p. 559.

[e] Larsen and Berman, ref. 19, pp. 224–225.

[f] Winchell, ref. 31, p. 368.

[g] Winchell, ref. 31, p. 264.

[h] Larsen and Berman, ref. 19, p. 228.

[i] Winchell, ref. 31, p. 63.

[j] Winchell, ref. 31, pp. 346–347.

[k] Winchell, ref. 31, pp. 376–380.

tory investigation of thin and polished sections. The study included a detailed logging of the 11,000 ft. of diamond-drill core from the National Lead Company's diamond-drilling program.

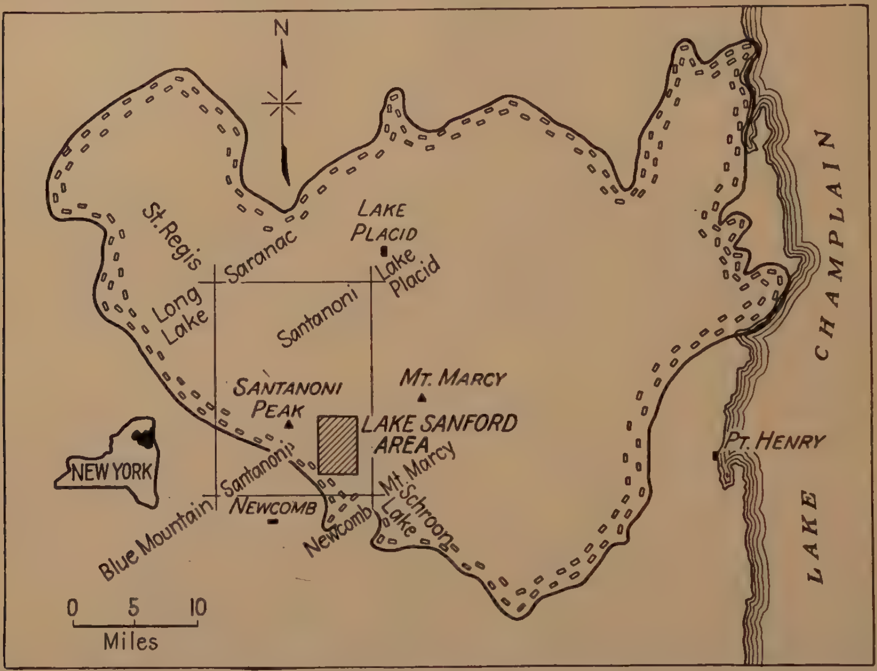

FIG. 1.—INDEX MAP SHOWING LAKE SANFORD AREA WITH REFERENCE TO THE ADIRONDACK MASSIF (*after Balk*).

## PETROGRAPHY

The names of the known ore bodies in the Sanford group are: the Sanford Hill, Ore Mountain, Calamity-Mill Pond, and Cheney Pond, and the location of these bodies is shown on the general geologic map of the area (Fig. 8). The ore bodies lie in the Hudson River valley, around Lake Sanford, within the core of the Adirondack anorthosite massif.

The anorthositic series in the Lake Sanford area consists of anorthosite, gabbroic anorthosite, gabbro, and titaniferous magnetite. These rocks represent a genetically gradational series and differ from one another primarily in the ratios of the constituent minerals. The characteristics of the minerals of the series are compiled in Table 1.

The divisions used in the mapping of the Sanford ore are as follows: anorthosite, 0 to 15 per cent mafic minerals; gabbroic anorthosite, 15 to 35 per cent mafic minerals and 50 to 15 per cent plagioclase phenocrysts; gabbro, 35 to 75 per cent mafic minerals, and not more than 15 per cent plagioclase phenocrysts; ore-bearing gabbro, 10 to 40 per cent ore minerals; gabbroic lean ore, 40 to 90 per cent ore minerals; and rich ore, 10 per cent or less gangue minerals. Buddington,[8] in 1939 set up a more detailed division for the Adirondack anorthosite series.

The anorthosite is a bluish gray to almost white, granular rock with a very coarse to medium grained texture. The coarse facies is composed of tabular labradorite phenocrysts in a groundmass of slightly more acid plagioclase (Fig. 4) and a small percentage of dark minerals. The phenocrysts in the coarse facies may be 10 cm. or more in length, but they average 4 cm. Minerals of the ground mass are 1 to 4 cm. long. The coarse facies may grade into medium grained anorthosite through a decrease in the number of plagioclase phenocrysts.

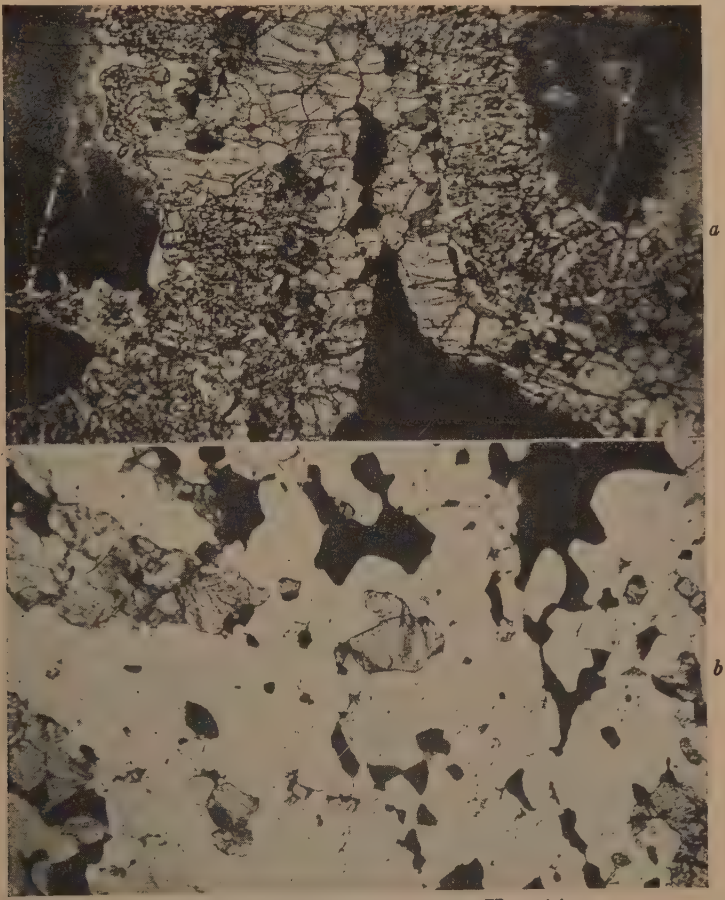

FIG. 2.—MINERALS FROM SANFORD HILL. × 40.

*a.* Reaction zone between ore and anorthosite.

Ore (black, lower center), is separated from labradorite (on either side), which is almost black, owing to dustlike inclusions it contains, by several zones of reaction minerals.

A very narrow zone of clear feldspar with minute spinel anhedra occurs between the inclusion-filled plagioclase and a band of vermicular garnet and feldspar. Between this zone and the ore is an augite band.

*b.* Fine-grained gabbro from diamond-drill core. Minerals are labradorite (light), augite (gray), and ore minerals (dark). Ore minerals have a very irregular anhedral shape.

The dark minerals are pyroxene, hornblende, garnet, and the ore minerals. Pyroxene and hornblende occur disseminated through the groundmass, while the ore and garnet form aggregates with ore surrounded by garnet. Apatite is an uncommon accessory mineral in the anorthosite. Miller[20] was one of the first to recognize that the mafic constituents of anorthosite may rise locally to as much as 25 per cent. The facies with more than 15 per cent of dark minerals is classed as gabbroic anorthosite in the Lake Sanford area.

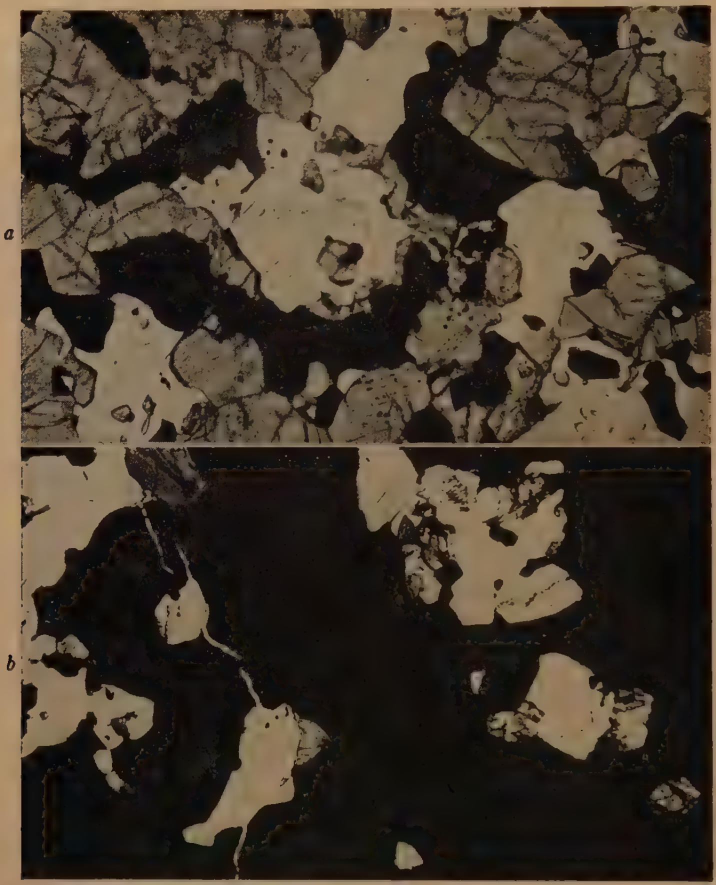

FIG. 3.—DIAMOND-DRILL CORE AT SANFORD HILL. × 40.
*a*. Ore-rich gabbro. Accompanying ore minerals are diallage, garnet, and labradorite.
*b*. Gabbroic lean ore. Narrow cracks (white) in ore (dark) are filled with spinel.

Much of the plagioclase is altered to scapolite, a characteristic high-temperature alteration product in gabbros.[25] Pyroxene and hornblende are commonly altered to chlorite, which is closely associated with calcite as late hydrothermal alteration products of the primary minerals.

The sequence of crystallization of the minerals in the anorthosite appears to have been as follows (earliest to latest): plagioclase, apatite, hypersthene, augite, hornblende, garnet, and ilmenite-magnetite. The large plagioclase phenocrysts are probably of intratelluric origin and were partially reworked by the feldspar-rich interstitial residuum. Peritectic reaction[1] between plagioclase and the ore residuum produced the garnet, and all or part of the hornblende was formed by hornblendization of the pyroxene.

The gabbroic anorthosite is a rock composed of 15 to 50 per cent labradorite phenocrysts in a medium grained groundmass ranging from 15 to 35 per cent mafic minerals. This rock type differs from anorthosite in having fewer plagioclase phenocrysts and more dark minerals. Gabbroic anorthosite is intermediate between anorthosite and gabbro, and this facies embraces all gradations between these two.

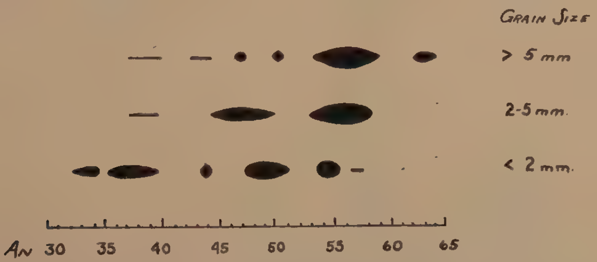

FIG. 4.—RANGE IN COMPOSITION OF PLAGIOCLASE GRAINS OF ANORTHOSITE AND GABBROIC ANORTHOSITE.

The percentage of anorthosite is represented horizontally, and the relative abundance of a given feldspar is shown by the thickness of the lines vertically.

Plagioclase compositions determined by use of Federov Universal stage.[10]

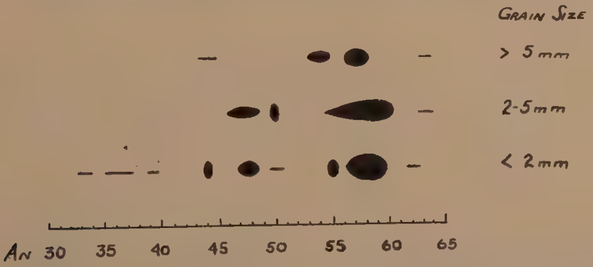

FIG. 5.—RANGE IN COMPOSITION OF PLAGIOCLASE GRAINS OF GABBRO.

Percentage of anorthite is represented horizontally, and the relative abundance of feldspar of a given composition is shown by the thickness of the lines vertically. (Plagioclase compositions determined by use of Federov Universal stage [Chudoba,[10] 1933].)

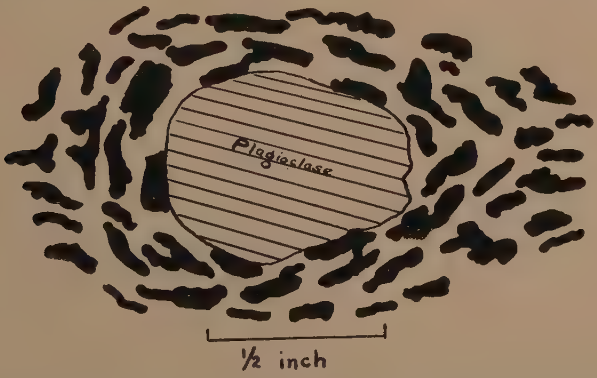

FIG. 6.—ORE MINERALS ARRANGED AS FLOW BANDS AROUND AN INCLUSION OF FELDSPAR.

Observed on a polished surface of ore. Magnetite, black; ilmenite, light.

The gabbro is a gray, medium to fine, equigranular rock, which may contain a few tabular plagioclase phenocrysts. The gabbro contains 35 to 75 per cent mafic minerals, and less than 15 per cent plagioclase phenocrysts. The gabbro may grade into anorthosite through gabbroic anorthosite, or it may intrude anorthosite. Through a decrease in the rock minerals and a corresponding increase in ilmenite and magnetite, there is a gradation from normal gabbro to ore-bearing gabbro to gabbroic lean ore to rich ore (Figs. 2*b* and 3).

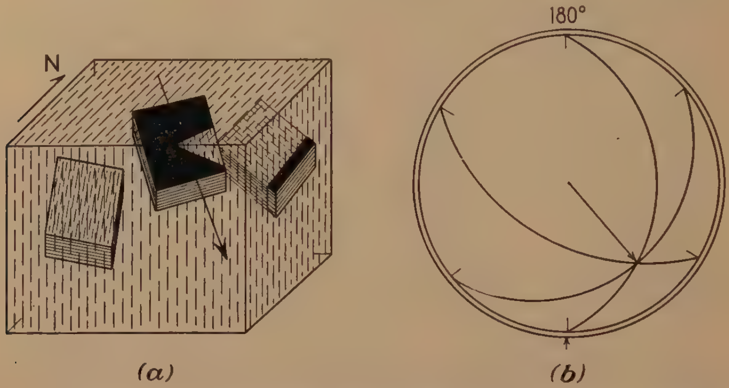

Fig. 7.—(*a*) Block diagram of anorthosite containing plagioclase phenocrysts with side pinacoids oriented around axis of flowage. (*b*) Traces of side pinacoid faces of plagioclase phenocrysts in *a* plotted on Schmidt equal-area net. Planes intersect in point that contains axis of flowage.

In Fig. 5 are compiled composition of plagioclase grains from the gabbro. There is not the marked difference in the composition of the large and small grains that there is in the anorthosite. The phenocrysts in the gabbro were probably few enough to be completely reworked as crystallization progressed. Blebs of recrystallized plagioclase of a more sodic composition are commonly found along albite twin planes of plagioclase in the gabbro.

All of the minerals found in the anorthosite are found in the gabbro, and in addition are found diallage, basaltic hornblende, biotite and spinel.

## Structural Geology

### *Structural Elements*

Planar flow structure in rocks is developed by an alignment of minerals during the period of flowage and crystallization in the magma. It records the probable direction of movement within the magma during this period.

The planar flow structure of the anorthosite of the Lake Sanford area is commonly very elusive, as pointed out by Balk (p. 315 of ref. 3) for the anorthosite of the Adirondack massif as a whole. Locally, however, the plagioclase phenocrysts orient with their side pinacoid (010) faces forming a planar flow structure. Such an orientation is very common in the gabbroic anorthosite. Gabbro usually shows a good primary foliation, which is accentuated by the dark minerals and ore minerals concentrating as bands or layers. Ilmenite and magnetite grains may show elongation into a planar flow structure parallel to contacts and inclusions as shown in Fig. 6.

The writer found a type of linear flow structure in much of the anorthosite that has not been described previously in the literature. It has been found that plagioclase phenocrysts are arranged with their side pinacoids (010) parallel to a common axis in much of the anorthosite (Fig. 7). Such an orientation indicates that the magma containing the suspended crystals moved with more or less uniform pressure exerted normal to the direction of flow, much as they would in a liquid moving through a pipe. The linear flow structures that indicate the direction of flow within the magma are shown on the geologic maps.

The Lake Sanford area is but a small portion of the Adirondack anorthosite massif, and for this reason a complete analysis of jointing is not possible. Systems of steeply dipping northeast-southwest, west northwest-east southeast, north-south and east-west regional joints predominate. Gabbro is typically much more jointed than anorthosite, which usually is massive. Several small diabase dikes follow the northeast-southwest joints.

Faulting is confined to minor post-ore displacements of a few inches along numerous joint surfaces of the ore and rock.

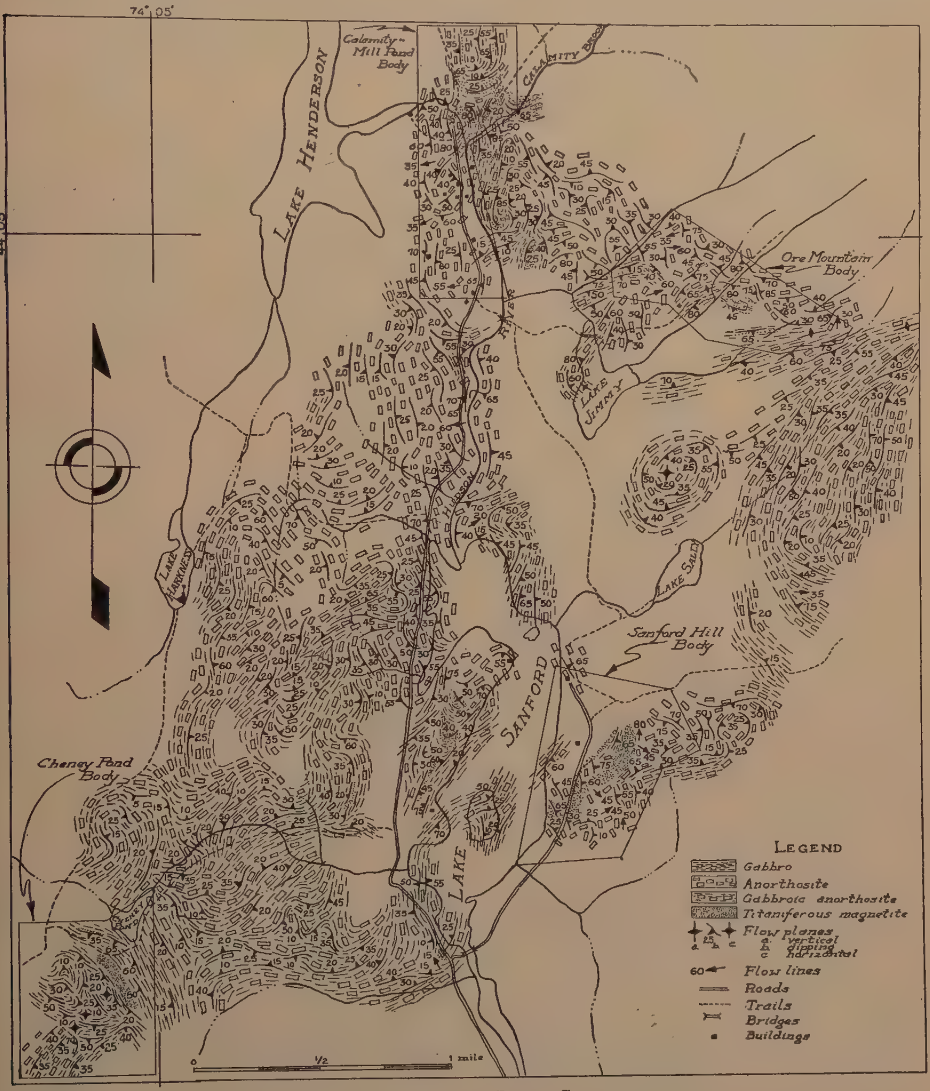

FIG. 8.—GEOLOGIC MAP OF LAKE SANFORD AREA.

### *Structure of the Area*

Fig. 8 is a geologic map of the Lake Sanford area, which shows the distribution of the rock types, and the flow structure pattern. Maps of the Sanford Hill (Fig. 9), Ore Mountain (Fig. 10), Calamity-Mill Pond (Fig. 11), and Cheney Pond (Fig. 12) ore bodies show in more detail the geology in the proximity of these ore bodies.

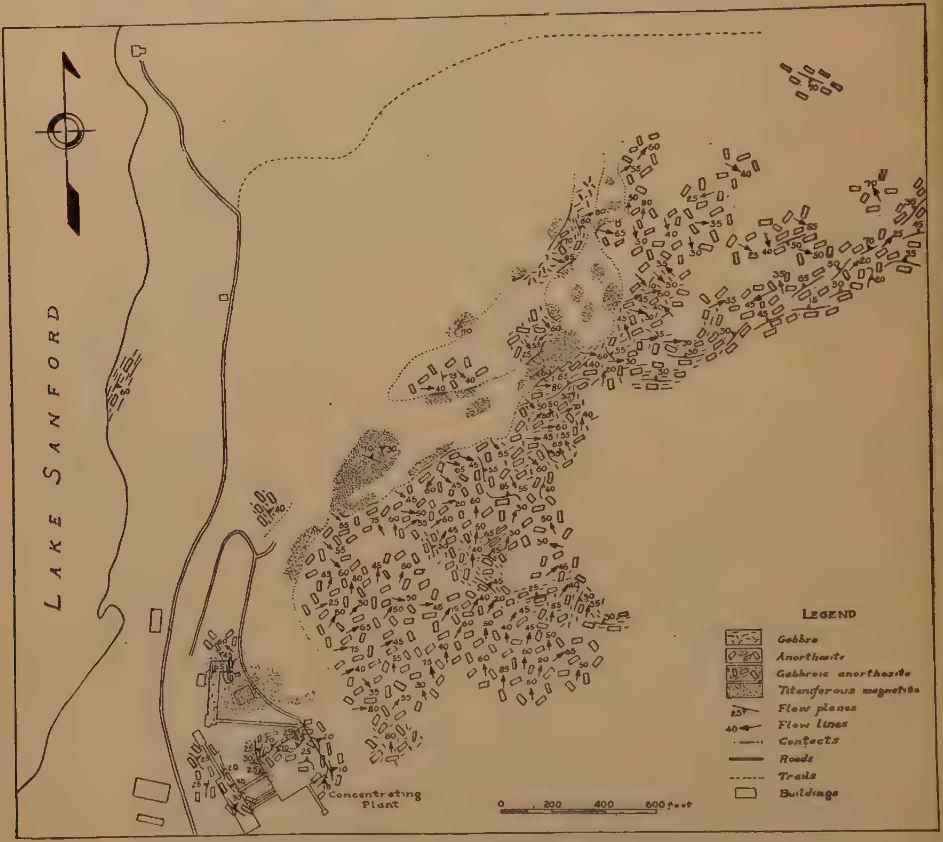

FIG. 9.—GEOLOGIC MAP OF SANFORD HILL ORE BODY.

The Sanford Hill, Calamity-Mill Pond, and Ore Mountain ore bodies occur in both anorthosite and gabbro, and the Cheney Pond ore body, which is much smaller, is a lens of ore in gabbro.

The largest of the ore bodies, the Sanford Hill, was very poorly exposed prior to the opening of the mine. The field study of this ore body was completed before stripping operations were started, but the author was permitted to study the 11,000 ft. of diamond-drill core. The ore in the Sanford Hill body occurs as bands, which grade into the gabbro, and in masses that replace anorthosite. The largest concentrations of ore occur in the anorthosite near the contact with the ore-bearing gabbro and at the contact. More recent observations since the mine benches have been developed bear out these earlier conclusions and indicate that the flow structures in the gabbro facies are roughly parallel to the contact with the anorthosite. Inclusions of anorthosite may be found surrounded by ore where the ore has replaced anorthosite, and locally ore has penetrated anorthosite along joint surfaces (Fig. 13*b*). Ore concentrations in the gabbro appear largest where there are bends in the flow structure.

The Ore Mountain ore body (Fig. 10) is similar to the Sanford Hill ore body in its

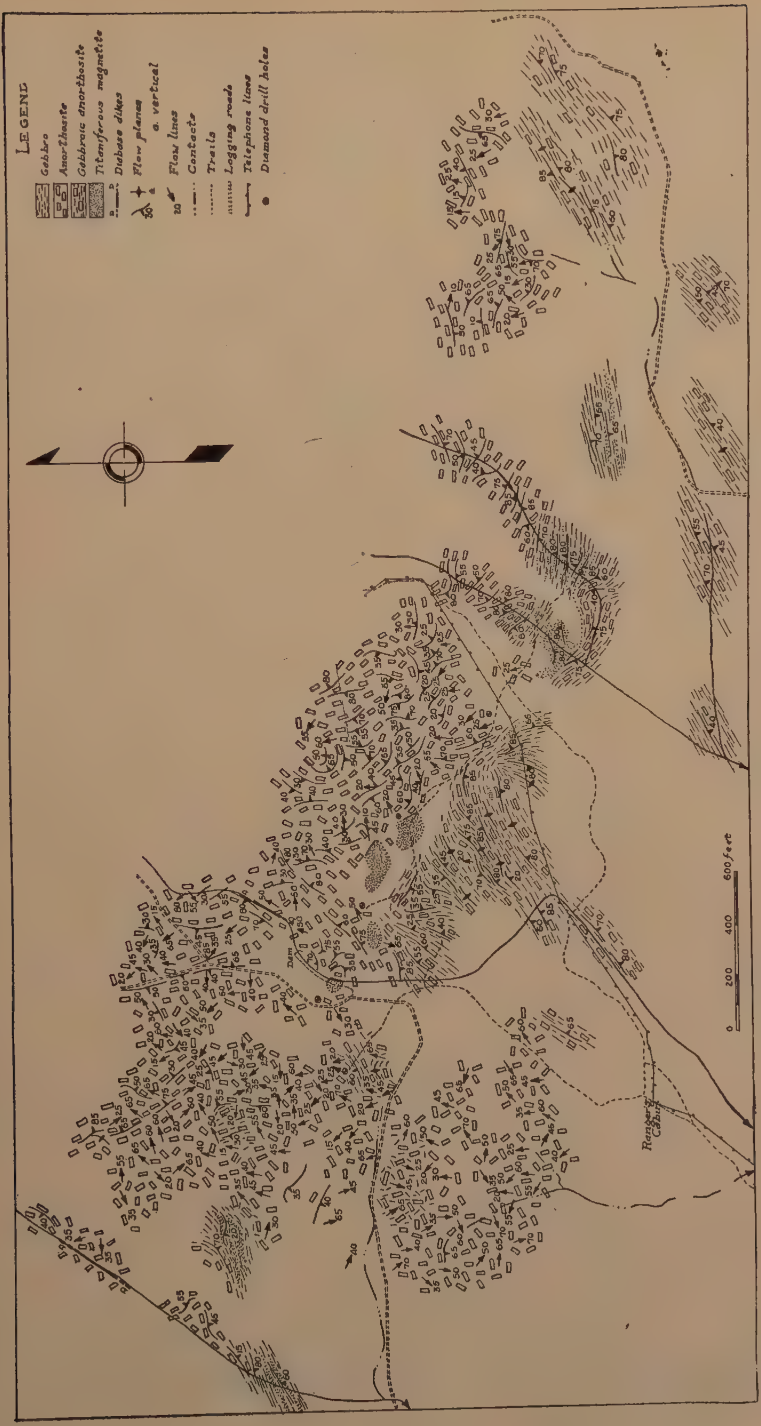

FIG. 10.—GEOLOGIC MAP OF ORE MOUNTAIN ORE BODY.

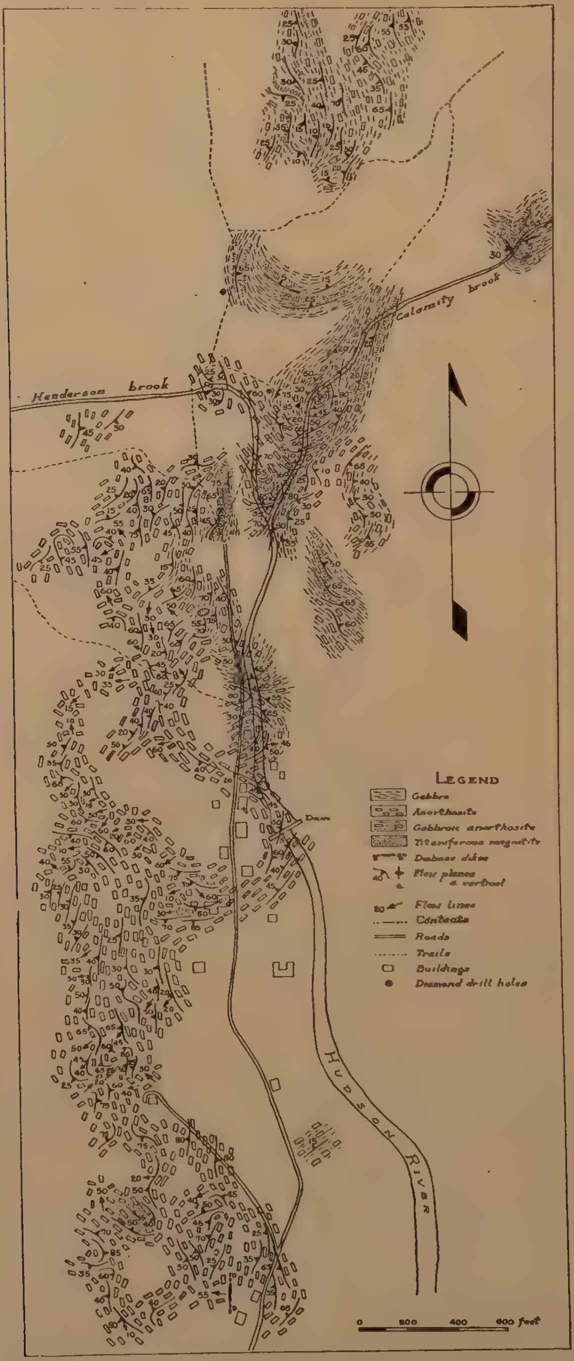

Fig. 11.—Geologic map of Calamity-Mill Pond ore body.

structural and petrographic relations. The outcropping ore is confined to a long, narrow zone characterized by anorthosite on one side and gabbroic facies on the other. Magnetic survey results indicate that the ore is restricted to the narrow zone at depth as well as at the surface.

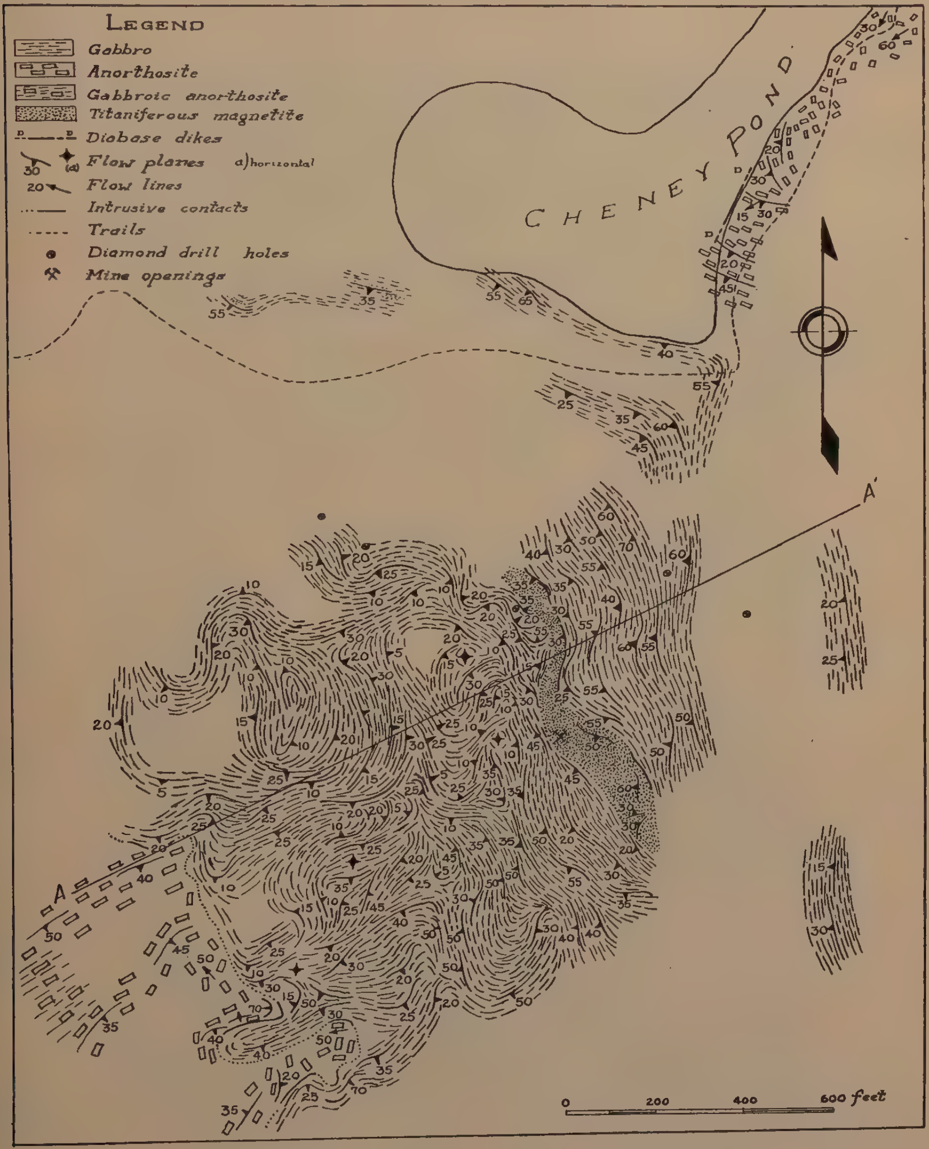

Fig. 12.—Geologic map of Cheney Pond ore body.

The Calamity-Mill Pond ore body is similar to the Sanford Hill and Ore Mountain bodies structurally and petrographically. There is very little ore associated with anorthosite, but most of the ore is associated with gabbro and anorthositic gabbro, which grade laterally into anorthosite. Magnetic survey results indicate that this ore body contains much less rich ore than the Sanford Hill and Ore Mountain bodies. Gabbroic lean ore was described by Newland[22] from prospect pits between Henderson and Calamity brooks. The structural relations as shown on the geologic map (Fig. 11) are much more complex than those suggested by Osborne (p. 740 of ref. 23).

The Cheney Pond ore body occurs as a small, lenticular mass parallel to the foliation of a gabbro body (Fig. 12). Fig. 14 shows the conformable nature of the ore in this gabbro, which is found intrusive into anorthosite to the southwest.

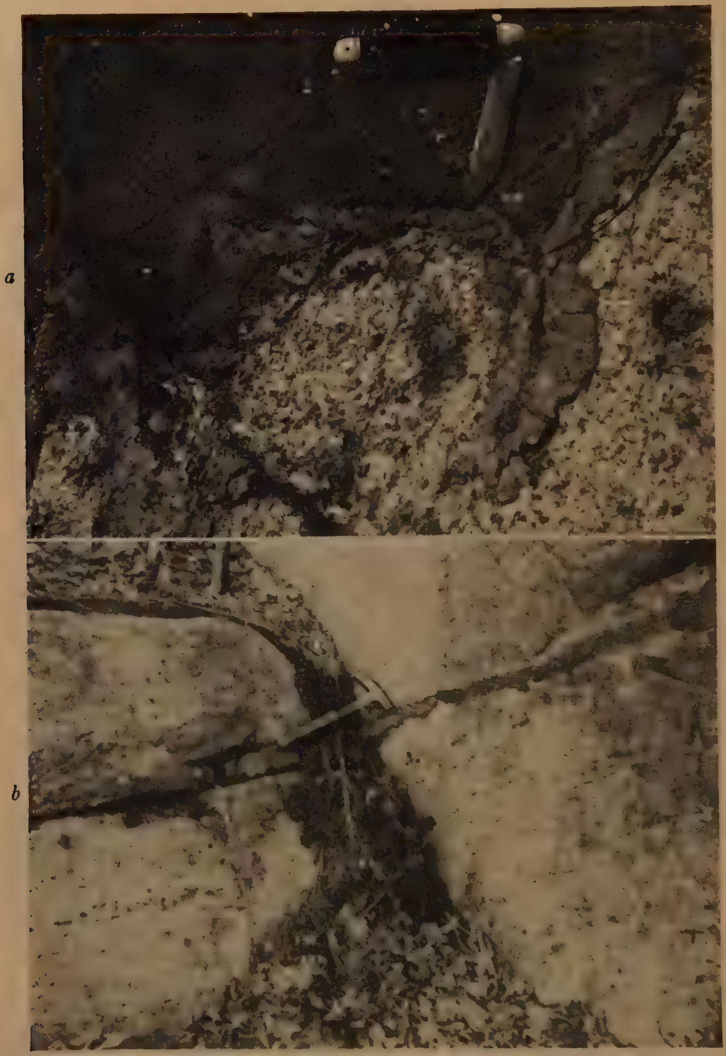

FIG. 13.—ORE AT SANFORD HILL.
*a*. Irregular contact of ore and anorthosite with zone of reaction minerals.
*b*. Stringer of ore along a joint in anorthosite.

## Petrologic History

The great irregularity in the plan of the flow structures of the Sanford area seems to bear out Balk's (p. 339 of ref. 3) assumption that it is in the apex region of the asymmetric intrusive dome. The parent magma from which the gabbro and anorthosite differentiated was probably of gabbroic anorthosite composition. Buddington[8] postulated a magma of this composition for parent magma of the whole Adirondack anorthosite series, but it is not the intention to suggest here that such a magma was common to all of the Adirondack massif.

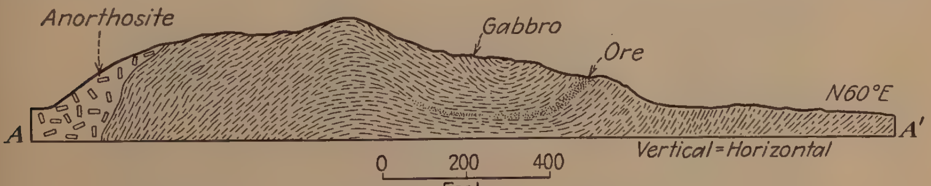

Fig. 14.—A cross section through Cheney Pond ore body.

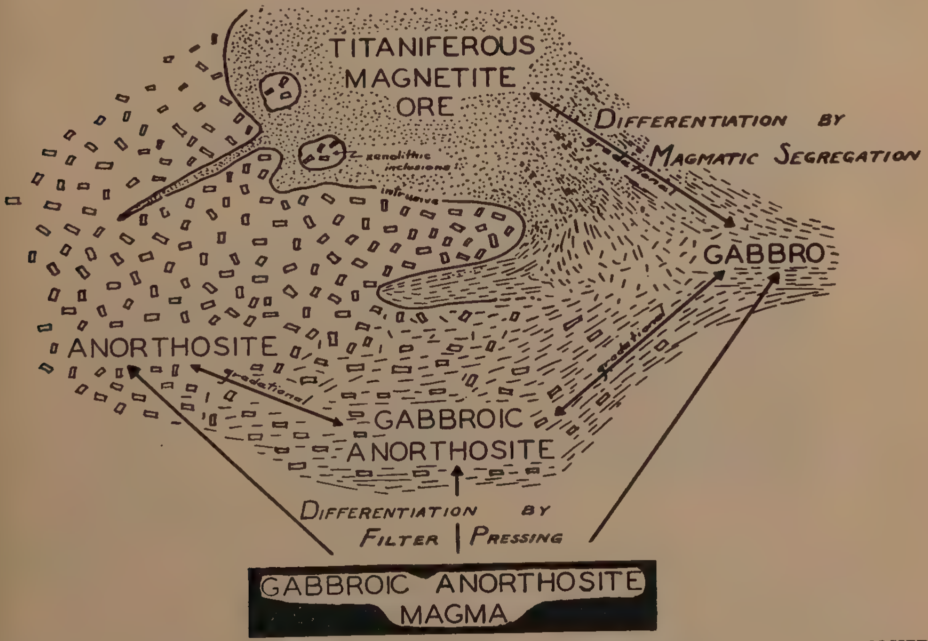
TITANIFEROUS MAGNETITE ORE
xenolithic inclusions
intrusive
Differentiation by Magmatic Segregation
gradational
GABBRO
ANORTHOSITE
gradational
GABBROIC ANORTHOSITE
Differentiation by Filter Pressing
GABBROIC ANORTHOSITE MAGMA

Fig. 15.—Petrogenetic diagram showing schematically the relations of the anorthosite, gabbro and titaniferous magnetite ore.

Phenocrysts of plagioclase of the anorthosite are of intratelluric origin and must have comprised at least one half of the magma prior to its intrusion, for they make up the bulk of the anorthosite. Compaction of the "mush" of crystals is evidenced by bent albite twins and protoclastic structure of the labradorite crystals in the anorthosite. Lagging behind, owing to difference of rate of flow of these phenocrysts during intrusion, effected an accumulation, undoubtedly aided by constrictions in the channelways followed by the intruding anorthosite. The mode of concentration, essentially filter-pressing,[2] yielded a titaniferous magnetite-rich gabbroic residuum in the Lake Sanford area, rather than a syenitic fraction as postulated by Balk for the massif as a whole (Fig. 15). Further differentiation within this gabbroic fraction probably gave rise to the concentration of titaniferous magnetite lenses in the gabbro.

The gabbro and anorthosite, after being separated by this mechanical method of differentiation, continued crystallizing in equilibrium with each other, as is indicated by the very slight differences in the plagioclase composition of the two rocks.

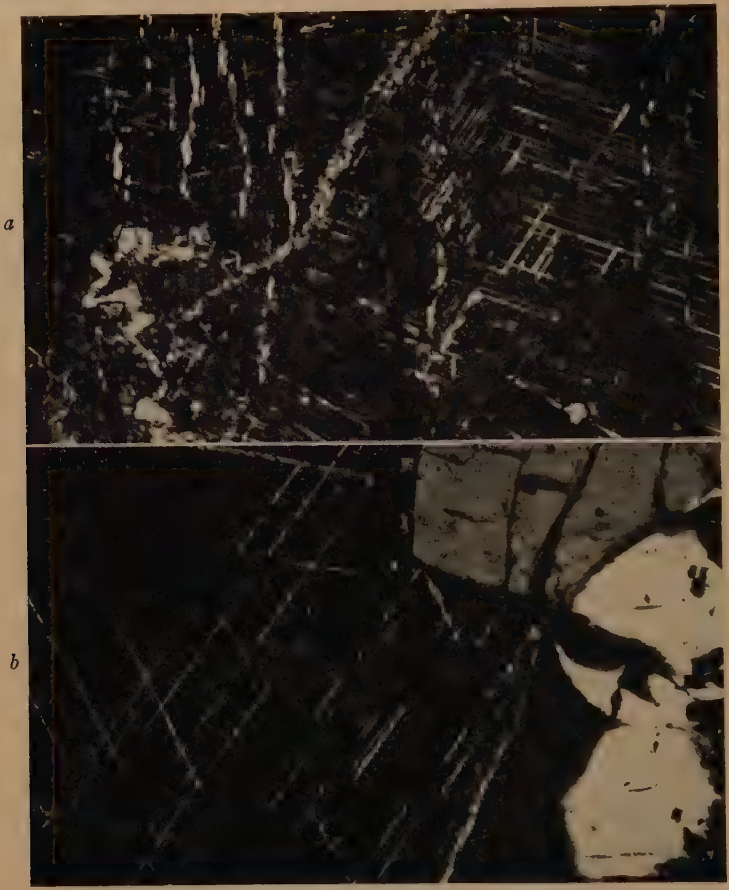

Fig. 16.—Ilmenite intergrowths. × 100. HCl etch.

*a.* Intergrowths of ilmenite (light) in magnetite (dark) from Calamity-Mill Pond body. Types of intergrowths are:

1. Regular tabular plates parallel to octahedral planes in magnetite.
2. Flecklike intergrowths parallel to octahedral planes in magnetite.
3. En échelon intergrowths.
4. Irregular concentrations along boundary of magnetite grains (runs diagonally across center of photomicrograph).

*b.* Ilmenite intergrowths (light) occur along octahedral planes of magnetite (dark) and along boundary of magnetite grains. Magnetite corrodes ilmenite grains. Garnet is subhedral against magnetite which penetrates along cracks in the garnet. From diamond-drill core at Sanford Hill.

The ore minerals are genetically related to the gabbro, and ore is not found anywhere in the Sanford area without this rock. Its concentration into bands in the gabbro was effected by fractional crystallization coupled with the difference in the rate of flow of this still mobile portion of the gabbro as compared to the already consolidated silicate minerals. Differentiation *in situ* could not have supplied all the ore. There must have been portions of the magma already rich in iron and titanium prior to intrusion, and ore deposition was focused at these points.

The anorthosite probably assumed the properties of a rigid body before the gabbroic anorthosite and gabbro, since it is composed of the early plagioclase phenocrysts. When the ore-rich gabbros encountered anorthosite masses, ore replaced anorthosite in a very irregular manner and penetrated it along already formed joints. This replacement of anorthosite by ore gave rise to the richest portions of the Sanford Hill and Ore Mountain ore bodies. There is not a definite sequence of chronologic units in the history of intrusion, but rather a series of overlapping relations.

## ORE DEPOSITS

### *Megascopic Description of the Ore*

Ore occurring in gabbro is hypidiomorphic, and the grain size varies from 1 to 2 mm. Ilmenite grains have a bright, metallic luster and irregular fracture; magnetite is dull and commonly shows a prominent octohedral parting. The ilmenite grains are frequently euhedral and subhedral. Gangue minerals of the same average grain size as the ore minerals occur disseminated through the ore, and these gangue minerals are the constituent minerals of the gabbro. The gradations between gabbro and ore have been discussed. The Cheney Pond ore is finer grained than ore found in gabbro elsewhere in the area, and it carries a higher percentage of rock minerals.

Concentration of the ore into bands that grade into gabbro suggests that the ore was formed *in situ.* Ore was formed in place, but there was also some movement of ore along these bands after the adjacent gabbro consolidated. Flow structures near the margins of bands of ore in gabbro are evidence of this movement.

The ore in anorthosite is hypidiomorphic and is slightly coarser than the ore in gabbro. The grain size is 2 to 3 mm. Physical differences of ilmenite and magnetite are essentially the same for this type of ore as for the ore in the gabbro, though euhedral grains of ilmenite in the magnetite are more abundant in this ore. Magnetite grains commonly appear to be subhedral.

Gangue of the ore in anorthosite consists of remnants of anorthosite ranging from single dark green feldspar phenocrysts to large blocks of the rock. These inclusions and ore-anorthosite contacts are always surrounded by a zone of reaction minerals of which garnet and pyroxene are the most conspicuous (Fig. 13*a*).

The ore may occur in large masses with little or no gangue. One diamond-drill hole on Sanford Hill cut through almost 200 ft. of solid ore. Other portions of the ore contain many individual plagioclase phenocrysts and xenoliths of anorthosite.

### *Microscopic Description of the Ore*

Table 2 summarizes data compiled from a study of polished surfaces of ore. Microscopically the ore in the gabbro is finer grained than the ore associated with anorthosite. The texture of ore is hypautomorphic-granular, except where it contains abundant gangue minerals; then the ore minerals are anhedral to the early formed gangue. Flow structures may be found in ore both in gabbro and anorthosite. The ore minerals in gabbro may be elongated parallel to the bands in which they occur, and they may be found forming flow bands around plagioclase inclusions (Fig. 6).

The ratio of $TiO_2$:Fe is much higher in gabbroic types of ore than in ore associated with anorthosite. In the gabbroic types the grains of ilmenite and magnetite are closely interlocked, but there are fewer intergrowths of ilmenite in magnetite. The ore

TABLE 2.—*Characteristics of Ore Specimens Studied under the Microscope by Use of Reflected Light*

| Ilmenite Intergrowths in Magnetite Per Cent | Relative Amount | | | Grain Size Average | | Important Structural and Textural Relations | Depth | Hole | Specimen |
|---|---|---|---|---|---|---|---|---|---|
| | Gangue | Magnetite | Ilmenite | Magnetite | Ilmenite | | | | |
| 5–7 | 25 % | 30 % | 45 % | 2.5 mm | 2.5 mm | Ilmenite grains subhedral, magnetite anhedral. Gangue clustered in patches. Magnetite penetrates ilmenite. | 131′6″ | 3 | 3g<br>Rich ore |
| 10 | 20 | 50 | 30 | | | Ilmenite grains subhedral, magnetite anhedral. Magnetite extensively embays ilmenite. Gangue clustered in patches. | 108′ | 4 | 4r<br>Rich ore |
| 5 | 20 | 45 | 35 | 2 | 3 | Ilmenite and magnetite subhedral. Magnetite embays ilmenite along cracks and boundaries. Gangue scattered. | 185′ | 4 | 4f′<br>Rich ore with garnet |
| 5 | 25 | 55 | 20 | | | Ilmenite grains subhedral, magnetite anhedral. Pronounced penetration of ilmenite by magnetite. Gangue scattered. | 117′ | 6 | 6x<br>Lean ore rich in dk mineral |
| 8–10 | 10 | 40 | 50 | 2 | 2.5 | Ilmenite and magnetite anhedral. Ilmenite grains corroded by magnetite. Gangue occurs in clusters. | 61′ | 8 | 8d<br>Rich ore |
| 5 | 7 | 33 | 60 | 1 | 1.5 | Ilmenite subhedral, magnetite anhedral. Magnetite penetrates ilmenite. Gangue occurs in rounded anhedra. | 165′ | 9 | 9b′<br>Fine-grained rich ore |
| 15–20 | tr | 55 | 45 | 1.5 | 1.5 | Ilmenite subhedral, magnetite anhedral. Magnetite embays ilmenite in an irregular manner. Veinlet of pyrite. | 28′ | 10 | 10c<br>Rich ore |
| 10 | 5 | 40 | 55 | 1.5 | 1.5 | Ilmenite and magnetite grains greatly elongated into a foliated structure. Boundaries of grains very ragged. | 5′ | 11 | 11b<br>Rich ore |
| 10 | 15 | 40 | 45 | 1.5 | 2 | Ilmenite subhedral, magnetite anhedral. Ilmenite embayed by magnetite. Gangue as anhedral masses in ore. | 64′ | 11 | 11l<br>Rich ore |
| 15–20 | 10 | 60 | 30 | 1.5 | 2 | Ilmenite subhedral, magnetite anhedral. Magnetite embays ilmenite. Gangue as scattered masses. | 76′ | 12 | 12e<br>Rich ore |
| 10–15 | 16 | 42 | 42 | 1.5 | 2 | Ilmenite subhedral, magnetite anhedral. Ilmenite only slightly corroded by magnetite. Gangue minerals irregular. | 140′ | 13 | 13w<br>Rich ore |
| 25 | 20 | 25 | 55 | 2 | 2.5 | Ilmenite and magnetite subhedral. Intergrowths of magnetite in ilmenite. Magnetite embays ilmenite. Gangue silicates irregular. | 36′ | 14 | 14c<br>Rich ore |
| 7 | 10 | 60 | 30 | 1.75 | 2 | Ilmenite and magnetite subhedral. Magnetite corrodes ilmenite. Just a few anhedral grains of gangue. | 12′ | 15 | 15a<br>Rich ore |
| 10 | 5 | 60 | 35 | 1 | 1.5 | Ilmenite subhedral, magnetite anhedral. Magnetite corrodes and penetrates ilmenite. Gangue intricately associated with ore. | 64′ | 16 | 16e<br>Rich ore |
| 8 | 10 | 70 | 20 | 2 | 2.5 | Ilmenite and magnetite subhedral. Magnetite corrodes ilmenite, penetrating it along cracks. Gangue largely late carbonate. | 167′ | 19 | 19e<br>Rich ore |
| 15–20 | 5 | 55 | 40 | 2 | 3 | Ilmenite and magnetite subhedral. Especially along cracks is ilmenite penetrated by magnetite. | 141′ | 22 | 22n<br>Rich ore |
| 10 | 40 | 25 | 35 | 1 | 1.5 | Ilmenite and magnetite anhedral—grains elongated into flow structure in some portions of polished surface. | 55′ | 27 | 27c<br>Fine-grained ore<br>Garnet-rich |
| 7–10 | 5 | 50 | 45 | 1 | 1 | Ilmenite subhedral, magnetite anhedral. Ilmenite grains rounded and embayed by magnetite. | 59′6″ | 30 | 30l<br>Rich ore |

TABLE 2.—(*Continued*)

| Ilmenite Intergrowths in Magnetite Per Cent | Relative Amount | | | Grain Size | | Important Structural and Textural Relations | Depth | Hole | Specimen |
|---|---|---|---|---|---|---|---|---|---|
| | Gangue | Magnetite | Ilmenite | Magnetite | Ilmenite | | | | |
| 8–10 | 5 | 55 | 40 | 1.5 | 1.5 | Ilmenite subhedral, magnetite anhedral. Ilmenite embayed by magnetite. Few anhedral grains of gangue silicates. | 86′ | 31 | 31m<br>Rich ore |
| 10 | 2 | 60 | 38 | 2 | 2.5 | Magnetite and ilmenite subhedral. Ilmenite penetrated by magnetite, especially along cracks. | 162′ | 31 | 31a′<br>Rich ore |
| | 40 | 20 | 40 | 1 | 1 | Ilmenite and magnetite anhedral. Grains arranged around an inclusion of gangue like wake around a moving boat. | 114′ | 32 | 32h<br>Lean ore |
| 15–20 | 10 | 60 | 30 | 1.5 | 2.5 | Ilmenite and magnetite anhedral. Ore minerals and gangue grains somewhat elongated into flow structure. | 215′ | 34 | 34y<br>Rich ore |
| 7–10 | 1 | 65 | 34 | 2.5 | 2 | Ilmenite and magnetite subhedral. Ilmenite very strongly corroded by magnetite. | 201′ | 35 | 35t<br>Rich ore |
| 15 | 5 | 65 | 30 | 1.5 | 2.5 | Ilmenite subhedral, magnetite anhedral. Ilmenite embayed by magnetite. | 271′ | 35 | 35b′<br>Rich ore |
| 7–10 | 10 | 40 | 50 | 1.5 | 2.5 | Ilmenite subhedral, magnetite anhedral. Quite intricate embayment of ilmenite by magnetite. | 181′ | 37 | 37p′<br>Rich ore |
| 5 | 15 | 60 | 25 | 1.5 | 2 | Ilmenite and magnetite subhedral. Corrosion of ilmenite by magnetite not intense. Gangue silicates scattered. | 500′ | 37 | 37f$^3$<br>Rich ore |
| 10–15 | tr | 75 | 25 | 2.5 | 3 | Ilmenite and magnetite subhedral. Ilmenite grains penetrated along cracks by magnetite. | 133′ | 38 | 38k<br>Rich ore |
| 5–10 | 40 | 35 | 25 | 1.5 | 2 | Ilmenite and magnetite anhedral. Corrosion of ilmenite by magnetite not strong. Feldspar remnants with reaction rims. | 102′ | 42 | 42h<br>Ore with anorthosite |
| 8 | 10 | 40 | 50 | 1 | 2 | Ilmenite subhedral, magnetite anhedral. Ilmenite-magnetite relations are mutual. Gangue as anhedra. | 62′ | 46 | 46b<br>Rich ore |
| 15 | tr | 40 | 60 | 2 | 2.5 | Ilmenite and magnetite are anhedral, being elongated into flow structure. | 42′ | 50 | 50n<br>Rich ore |
| 5–7 | 15 | 35 | 50 | 1 | 1.5 | Ilmenite subhedral, magnetite anhedral. Corrosion of ilmenite by magnetite especially strong here. | 76′ | 51 | 51o<br>Rich ore |
| | 10 | 35 | 55 | 1.5 | 2 | Ilmenite and magnetite anhedral. Magnetite replaces ilmenite in a very irregular manner. Elongated grains. | 51′ | 52 | 52p<br>Fine-grained rich ore |
| 15–20 | 5 | 55 | 40 | 2 | 2.5 | Ilmenite and magnetite anhedral. Magnetite embays ilmenite especially along cracks. | 11′ | 53 | 53a<br>Rich ore |
| 3–5 | 20 | 50 | 30 | 2 | 2.5 | Ilmenite and magnetite subhedral. Ilmenite penetrated by magnetite especially along cracks. Gangue irregularly distributed. | 114′ | 54 | 54b′<br>Rich ore |
| 10–15 | 25<br>(pyrite) | 40 | 35 | 3 | 3.5 | Ilmenite and magnetite subhedral. Ilmenite embayed by magnetite. Pyrite vein 2–7 mm. wide cuts ore minerals | 186′ | 54 | 54r′<br>Rich ore with pyrite stringer |
| 4–5 | 15 | 30 | 55 | 1 | 2 | Ilmenite and magnetite anhedral; grains oriented and elongated parallel the flow structure. Gangue as many anhedra. | 54′ | 63 | 63f<br>Lean ore |
| | 10 | 40 | 50 | 2 | 3.5 | Ilmenite subhedral, magnetite anhedral. Magnetite corrodes ilmenite along cracks and boundaries. | 55′ | 47 | 66i<br>Rich ore |
| 7–10 | 25 | 40 | 35 | 1.5 | 2 | Ilmenite anhedral—as is magnetite. Gangue silicates scattered through ore, giving ore minerals anhedral shape. | ? | ? | ?<br>Lean ore |

in anorthosite is rich in intergrowths of ilmenite in magnetite, of which there are five types:

1. Regular tabular plates of ilmenite parallel to octohedral planes of magnetite (Figs. 16, 17*b*, 18*a*).
2. Irregular tabular plates parallel to octohedral planes of magnetite (Figs. 17*b*, 18*b*).
3. Minute flecklike intergrowths parallel to octohedral planes of magnetite (Figs. 16*a*, 17*b*).
4. En échelon intergrowths (Fig. 16*a*).
5. Irregular concentrations between magnetite grains (Fig. 16).

The intergrowths of type 1 are by far the most common. It is now generally agreed that these intergrowths are formed by unmixing (Bastin et al.[5]) of ilmenite from magnetite at a temperature between 500° and 800°C.[24] or higher.[17] Grüner[18] concluded that ilmenite and magnetite share oxygen planes of their atomic structures to form the typical octohedral exsolution

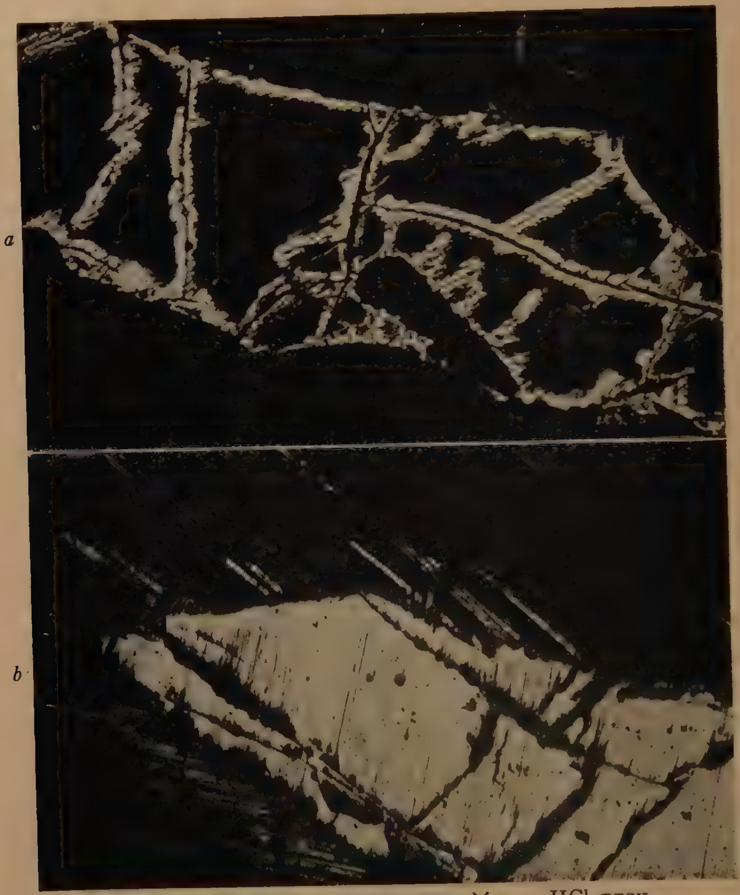

FIG. 17.—ILMENITE IN MAGNETITE. × 100. HCl ETCH.
*a*. Relic of ilmenite (light) in magnetite (dark) from Cheney Pond body.
*b*. Ilmenite (light) corroded along boundaries by magnetite (dark), which also penetrates along cracks in ilmenite. Ilmenite lamellae in magnetite are more abundant near ilmenite grains. Diamond-drill core at Sanford Hill.

intergrowths. En échelon arrangement of intergrowths of ilmenite are described by Faessler and Schwartz[13] as being caused by crystallographic slippage or translation during exsolution.

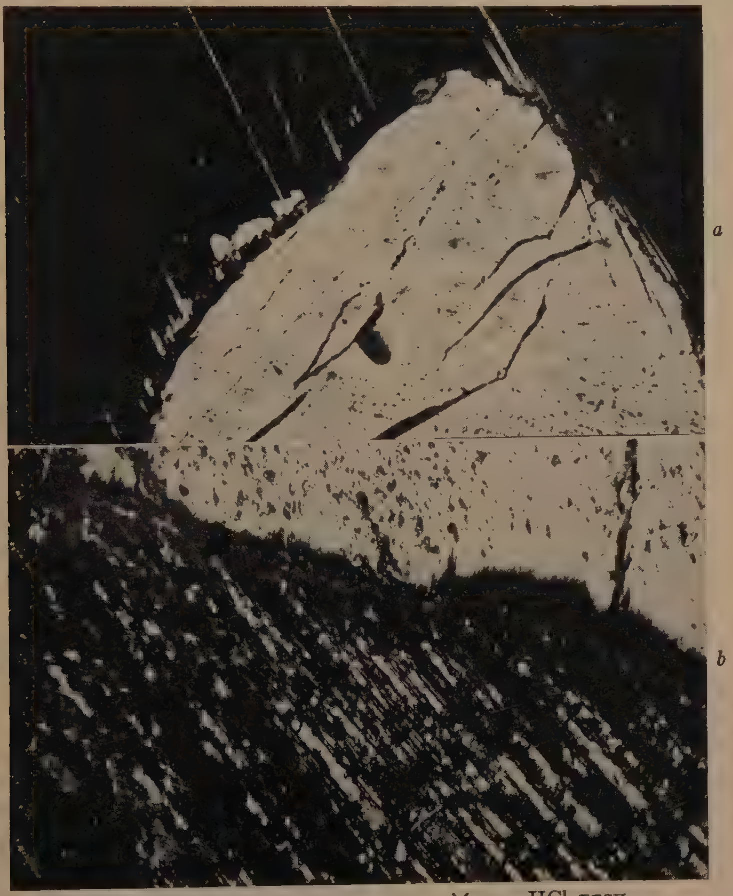

FIG. 18.—ILMENITE IN MAGNETITE. × 100. HCl ETCH.

*a.* Ilmenite (light) corroded by magnetite (dark). A few ilmenite lamellae in magnetite. Diamond-drill core at Sanford Hill.

*b.* Ilmenite (light) embayed along cracks by magnetite (dark). Irregular intergrowths of ilmenite closely spaced in magnetite.

Inclusions of ilmenite in magnetite are common. These are characterized by ragged borders, which suggest ilmenite replaced by magnetite. Relics of ilmenite in magnetite (Fig. 17*a*) are found occasionally.

An average of 0.50 per cent $V_2O_5$ has been reported by the National Lead Co. in numerous analyses of Sanford ore. The vanadium occurs in the magnetite, but beyond this nothing is known about its mode of occurrence. Dunn[11] recognized a vanadiferous maghemite, which he called coulsonite, in titaniferous iron ores in India. He assigned to it the tentative formula $FeO \cdot (Fe, V)_2O_3$. No coulsonite was found in any of the Sanford ores, nor were any other vanadium minerals found. It is very likely that the vanadium is contained in solid solution in the magnetite.

Gangue sulphides, pyrrhotite and pyrite, are present in small amounts as rounded anhedra. Pyrite also occurs as very narrow veinlets cutting both ore and rock minerals. This pyrite was probably the last mineral of the magmatic sequence to crystallize.

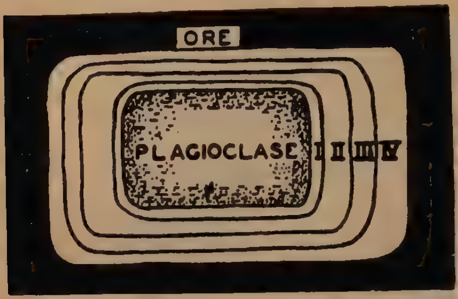

FIG. 19.—SCHEMATIC DIAGRAM OF REACTION ZONE BETWEEN ORE AND ANORTHOSITE.
Zone I, clear feldspar and small spinel anhedra; II, garnet; III, augite, with some green hornblende; IV, hypersthene, diallage, brown hornblende and biotite.

Gangue silicates occurring in the gabbroic types of ore have already been discussed, being the constituent minerals of the gabbro. The gangue of the ore in the anorthosite consists of remnants of anorthosite and individual labradorite crystals left also as remnants in the replacement of anorthosite by ore. The phenocrysts of plagioclase found in the ore are dark green megascopically, and many are nearly opaque in transmitted light, because of dustlike and rodlike inclusions of magnetite lying parallel along the albite twin planes of the plagioclase. With all occurrences of ore associated with anorthosite, there are typically zones of reaction minerals between the ore and rock, varying from a few millimeters in width up to 5 or 6 cm. (Fig. 3*a*). Fig. 19 is a schematic representation of the reaction zone. The minerals of the reaction zones vary greatly in abundance, but garnet is the commonest mineral. These minerals formed by deuteric reaction between the ore-bearing residuum and the already consolidated anorthosite.

### *Origin*

Ore bodies of magmatic origin were recognized by only a few investigators prior to Vogt's classic paper,[30] which firmly established the concept of accumulation of pyrogenetic minerals to form ore bodies. Beyschlag, Vogt, and Krusch[7] concluded, as many others, that the ore minerals crystallized early, aggregated, and were resorbed to be intruded as a magma. Tolman and Rogers[29] conclusively proved that magmatic sulphides replaced silicates after the consolidation of the igneous rocks and emphasized the role of mineralizers in the process. Singewald[27] pointed out that similar conditions existed between oxidic ores and primary silicates. Newhouse,[21] through an exhaustive study of opaque oxides and sulphides of rocks, has given conclusive evidence that ore minerals crystallize later than common rock minerals of basic rocks in many cases. There is little doubt that the titaniferous magnetite in the Sanford area came at the end of the crystallization sequence of the igneous rocks.

Bateman[6] recently proposed a classification of magmatic deposits, as follows:

I. Early Magmatic
    A. Disseminated
    B. Segregation
    C. Injection
II. Late Magmatic
    A. Residual liquid segregation
    B. Residual liquid injection
    C. Immiscible liquid segregation
    D. Immiscible liquid injection

Early magmatic deposits are those formed during the main stages of magma crystallization and embrace those deposits termed orthomagmatic and orthotectic. Late magmatic deposits are those formed from pyrogenetic minerals toward the close of the magmatic period. Bateman classes Adirondack titaniferous ores as residual liquid injections on the basis of Osborne's study.

Osborne (p. 735 of ref. 23) outlined the following classification for titaniferous deposits of the Adirondacks, Quebec, and Ontario:

1. Discordant or dikelike bodies
   - A. In anorthosite
   - B. In gabbro
   - C. In other rocks
2. Concordant, sill-like, or stratiform bodies
   - A. In anorthosite
   - B. In gabbro
   - C. In other rocks

The deposits of the Lake Sanford area belong to Osborne's types 1*A* and 2*B*. The Sanford Hill, Ore Mountain, and Calamity-Mill Pond ore bodies include both types. The discordant bodies in anorthosite are related to adjacent concordant bodies in gabbro, for the ore concentrated in gabbro prior to introduction into anorthosite. For this reason it is not uncommon to find concordant lenses of ore in gabbro within a few feet of discordant ore in anorthosite. Osborne did not recognize the gradation between anorthosite and gabbro which contains concordant lenses of ore. He classes the ore bodies of such a character in the Sanford area as concordant bodies in anorthosite. Osborne (p. 895–899 of ref. 23) erroneously classes the Cheney Pond ore body, which is conformable in gabbro, as a concordant body in anorthosite.

Oxide ores comparable to Beyschlag, Vogt, and Krusch's injection sulphide deposits are, according to Singewald,[28] the schlierenlike injections and dikes of ore that represent liquid ore differentiates squeezed into new positions in either still molten or already consolidated portions of the magma.

Osborne (pp. 730–732 of ref. 23) proposed the term "magmatic injection" for intrusive deposits of pyrogenetic minerals. He suggested this term to describe some titaniferous-magnetite ore bodies that cannot be termed magmatic segregations because they are not concentrations of the early crystallizing minerals in place. Osborne would restrict magmatic segregation to describe deposits in which there is a concentration of earlier minerals along margins.

Osborne proposed filter-pressing of anorthosite to yield a magma containing pyroxene, plagioclase and iron ore. This rest magma was then injected into anorthosite to form ore bodies and gabbro dikes. The writer agrees with Osborne's fundamental concepts of genesis of the Sanford ore, but would suggest some modifications in view of additional structural and petrographic data.

Conformable lenses of ore in gabbro were not recognized by Osborne, nor were rapid gradations from anorthosite to gabbro and ore, such as occur along Calamity Brook. Lenses of ore in gabbro are associated with all the ore bodies in the Lake Sanford area. Petrographic and structural evidence strongly support the conclusion that the ore concentrated as a late liquid residuum in the gabbro. This ore-rich residuum formed lenses, which grade into the enclosing gabbro. These lenses are magmatic segregations. Foliation of ore minerals along the boundary of some of these lenses indicates that the ore residuum flowed through the lenses as crystallization of ore minerals was in progress.

Parts of the anorthosite in the areas of ore deposition were not completely solidified at the time of the segregation of the ore from the gabbro. The anorthosite in these parts consisted of plagioclase phenocrysts in an unconsolidated groundmass. As the ore residuum segregated from the gabbro, it incorporated some of the phenocrysts of plagioclase from the unsolidified rock. At numerous points in the Sanford Hill ore body can be found ore containing little or no gangue minerals other than numerous, isolated plagioclase phenocrysts, which are frequently surrounded by flow-banded ore. The author believes that these occurrences were formed by injection of yet unconsolidated anorthosite by ore. It is difficult to evaluate the importance of such a process as opposed to replacement of anorthosite by ore, but replacement was probably far more widespread and important.

The ore-bearing gabbro grades into anorthosite and gabbroic anorthosite in the Sanford Hill, Ore Mountain, and Calamity-Mill Pond ore bodies. Where ore moving along encountered already consolidated anorthosite, it penetrated and replaced this rock to form discordant bodies. Osborne would call these magmatic injections. Localization of ore of this type forms large masses in the Sanford Hill and Ore Mountain ore bodies. That the ore residuum was tenuous is indicated by its ability to penetrate anorthosite out from the contact in an irregular manner and also as well-defined stringers, which occur along joint planes. Singewald[27] suggested that such evidence was indicative of participation of mineralizers in magmatic ore deposition. The abundance of pyroxene compared with the supply of hornblende and biotite in the reaction zone between anorthosite and ore indicates that volatile constituents entered only to a minor extent into the formation of the reaction minerals.

The writer concludes that the Lake Sanford titaniferous iron ores are magmatic segregations in gabbro and magmatic injections and replacements in anorthosite. The ore residuum of gabbro supplied the ore constituents, which form large masses in anorthosite. The magmatic segregations in gabbro would conform to Bateman's type 11*A*, residual liquid segregation.

As has already been pointed out, the importance of replacement as a process of ore emplacement is difficult to evaluate. Reaction rims between ore and anorthosite are always found, and xenoliths of anorthosite may be observed at all stages of replacement by the ore minerals. It is possible that reaction occurred between the ore and anorthosite only in final stages of a residual liquid ore injection, but it seems more likely that replacement was the dominant process for the introduction of ore into anorthosite.

## Acknowledgments

The study of the Lake Sanford area was undertaken with the cooperation and assistance of the New York State Museum. The writer is indebted to Dr. C. E. Guthe, Director of the New York State Museum, and Mr. C. A. Hartnagel, New York State Geologist, for permission to publish this report. The National Lead Co., Titanium Division, was very helpful in making information available. The faculty of the Johns Hopkins University; Prof. W. H. Newhouse, M.I.T., and Dr. A. H. Hagner, University of Wyoming, made many helpful suggestions in the preparation of the manuscript.

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# The Iron Deposits of Larap, Philippine Islands

By F. H. Kihlstedt,* Associate Member A.I.M.E.

(Chicago Meeting, February 1946)

## Abstract

The Larap iron deposits, 125 miles east of Manila, are the biggest high-grade iron deposits in the Philippines, and have in seven years produced nearly 4 million tons of 60 per cent ore. Magnetic surveying followed by churn and diamond drilling cheaply developed a score of million tons. The ore zone is a wedge of metamorphic rocks in fresh, Miocene sediments overlain by agglomerates, which seem to have filled a river valley eroded in the sediments. A mild overthrust of the competent agglomerates crumbled the shale in front. Solutions gradually turned it into more competent hornfels and deposited pyrite, then magnetite and pyrite in the pores. As the temperature rose large bodies of magnetite with some pyrite were deposited by replacement and in open cavities in tensional areas of overthrust shears. At still higher temperature magnetite and calcium-magnesium silicates formed other bodies in similar cavities, and finally the thrust zone was invaded by syenite porphyry, causing high-temperature alteration products. Possibly, as temperature abated, more high-grade ore was deposited. The complicated faulting indicates dynamic, chronological and spatial changes of mineralization rather than zoning in a static system. Noncommercial copper, molybdenum and some gold accompanied the iron mineralization.

In the thirties the Philippines gained a justified reputation as a country of considerable mineral wealth. Traces of Chinese workings show that these earliest travelers and traders had in a small way exploited the latent gold resources in the Islands. The Spaniards continued this for centuries, but did not succeed in establishing any steady mining industry. Modern technology was required to get it started.

The rapid development in the last decade before the war got its impetus from several financial factors, the return to sugar growers of previously collected taxes, the influx of Spanish capital during the civil war in Spain, the influx of Chinese capital when the Japanese started their "China incident," and others. Directly or indirectly much of this capital reached the mining industry; in particular, gold mining. Out of the turmoil of an exceptional mining boom emerged a sufficient number of good and well managed gold properties to send the production to a level above that of any single state in the United States at the time. Other ore deposits besides those of gold also received their share, and for a country without any heavy industries at

Table 1.—*Philippine Mine Output in* 1939

| Metal or Ore | Quantity | Value in Pesos |
|---|---|---|
| Gold | 1,033,037 oz. | 72,175,682 |
| Iron ore | 1,166,781 long tons | 4,914,800 |
| Chromite | 132,177 long tons | 2,295,167 |
| Copper concentrates | 12,093,670 lb. | 2,236,393 |
| Silver | 1,349,454 oz. | 1,752,591 |
| Copper ore | 25,333 long tons | 726,091 |
| Manganese ore | 29,394 long tons | 602,623 |
| Lead | 88,901 lb. | 6,781 |
| Total |  | 84,710,128 |

[a] Philippine Mining Yearbook, 1940.

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Manuscript received at the office of the Institute Nov. 7, 1945. Issued as TP 2001 in Mining Technology, May 1946.

* Atlantic, Gulf and Pacific Company of Manila; formerly Superintendent, Philippine Iron Mines, Inc.

all the mine output had in 1939 reached the impressive total of 42 million dollars (Table 1).

Next to gold, iron and ferroalloy metals dominated the production picture with a total of ₱7,812,590 in iron ore, chromite and manganese ore. From observations, the author believes that other ores of this type will be found in commercial quantities in the future.

Conspicuous for their absence are fuels. Although some tertiary coal has been mined airly steadily it is of low quality and used only for steam boilers and similar purposes. The small deposits of coking coal that have been found have not proved their worth. Attempts to develop oil deposits have failed so far, but the asphalt deposits on Leyte show that oil is, or at least once was, present.

Cheap hydroelectric power is not available. Favorable power sites, particularly in Mindanao, might some time in the future serve a heavy industry with power.

Thus, although the ferro-type ores are there they cannot be utilized within the country. Before the war they were exported. This limited exploitation of iron ore, chromite and manganese ore to the highest grades. Conditions permitting a lowering of grade would add enormously to the reserves of these ores.

This is particularly true of iron ores. The potential reserves of chrome-nickel-bearing residual ores in northeast Mindanao and adjacent islands are measured in many hundreds of millions of tons. Oölitic ores in the tertiary sedimentary complex have the same possibilities as similar ores elsewhere in the world. The author has seen magmatic ores both with and without titanium, as well as metamorphic quartz-banded hematites of some consequence.

However, the only ores mined have been the high-grade, low-sulphur parts of deposits belonging to the ill-defined class of hydrothermal or replacement ores, whose weathered outcrops of hard hematite, in place or in boulders, constitute some of the highest quality iron ores in the world. As they are very conspicuous in the landscape, they attracted the attention of early explorers and prospectors.

The often prominent boulder accumulations are, more often than not, deceiving. The hematite is a leached oxidation product of magnetite of unknown quality. The boulders may represent the whole ore reserve, as rapid, tropical erosion may have lowered the ground level to the roots of the ore deposit, or they may cover a large area out of all proportion to the size of the deposit.

Magnetic surveying, followed by drilling, has been found necessary as the first step of exploration. For further intelligent development and planning an early recognition of the master structure and the technique of ore deposition will save much money that otherwise will be spent wandering in the dark.

The ores in the same deposit may partly fill structural cavities and partly replace country rock, even where such commonly replaced rocks as limestone are present. The structure may be related to tension fracturing, saddle reefs, thrust faulting, etc., with greatly varying effect on shape and depth of ore bodies. In a structure that has caused extensive tensile conditions, the ore deposit is likely to be large and can be cheaply mined, and vice versa.

The Larap area is typical of Philippine magnetite deposits and its geology is very instructive from many viewpoints.

## LOCATION AND HISTORY OF LARAP AREA

The Larap mine was the first and to date is the largest iron mine of the district to be brought into production. It is on the east coast of Luzon, about 110 miles southeast of Manila. It is at the western end of the Paracale gold district, the second largest producer of gold in the Islands.

The conspicuous outcrops of hematite have been known for a long time and were described by the early geologists of the Philippine Bureau of Mines.[1] Tegengren did considerable exploration work, of which a description was published in 1927.[2] He returned in 1931 for another examination, and the present author briefly described it in 1940.[3]

Prior to 1930, attempts to open these mines failed because of trade difficulties, although in 1919 a Japanese company mined about 50,000 tons from the northern end of the deposit on the island of Calambayungan. The present company, Philippine Iron Mines, Inc., took over the claims in 1930. An estimate of some 6 million tons of ore was made by Tegengren, and on his findings the company, under the management of Atlantic, Gulf and Pacific Co. of Manila, began production late in 1934. Up to the outbreak of the war in the Pacific production was as follows:

| Year | Production, Long Tons |
|---|---|
| 1934 | 13,302 |
| 1935 | 305,611 |
| 1936 | 594,975 |
| 1937 | 592,482 |
| 1938 | 768,365 |
| 1939 | 706,563 |
| 1940 | 524,234 |
| 1941 | 436,149 |
| Total | 3,941,681 |

Thus a tonnage equal to two thirds of the original estimate of ore has been mined. Present ore reserves are estimated at a score of millions of tons, including some high-sulphur ore, which was added to the reserves by drilling in 1939 and 1940. Considerable sections of the ore-bearing zone remain unexplored.

## EXPLORATION

During the first year of production, hematite alone was mined. Only in 1936 was magnetite accidentally discovered in a hole drilled for water. Subsequently the whole area was covered by magnetic prospecting with a Tiberg magnetometer.

Both vertical and horizontal components were determined. Theoretical considerations pointed to the horizontal component as the more useful, because the earth's own field is nearly horizontal in the Philippines, and the horizontal component gave so much sharper pictures of the apparent widespread mineralization underground that determination of the vertical component was dropped. A study of the magnetic map in relation to the outcrops left little doubt that the hematite was a superficial oxidation product of magnetite ore bodies.

The magnetic survey was followed up by 10 diamond-drill holes. They all cut ore and showed that the ore formation dipped flatly, so that further exploration could be done with vertical holes. For some distance below the outcrops the rocks were softened by decomposition and the ore itself was of medium hardness. All this pointed to churn drilling as the logical method of exploration.

When drilling had reached below sea level, certain areas of the hanging wall were found to be too hard for economical churn drilling, and two diamond drills were acquired to complete unfinished holes. The combination of churn and diamond drilling then became standard practice. A churn drill would be taken off blast-hole drilling for a shift or two in order to drill down to the fresh, hard, hanging wall. Casing was left in the hole until a diamond drill had completed the hole. Not only did this system prove cheap, but the information on depth of decomposition helped enormously in laying out pit limits and planning underground mining.

At the height of the exploration campaign three 6-in. Bucyrus-Erie churn drills and two Boyle Brothers diamond drills were working, with ready-set diamond-drill bits supplied by the latter company.

During 1940 exploration data were as follows:

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[1] References are at the end of the paper.

| Type of Drilling | Feet | Cost per Foot |
|---|---|---|
| Churn | 45,668 | $0.24 |
| Diamond | 11,940 | 1.26 |
| Combined | 57,608 | 0.45 |

With 13,400,000 tons of ore drilled during the year, the cost of exploration per ton was $0.002. This low figure permitted much detailed drilling on the complicated ore body, slated to be mined first. As the political skies began to darken, this detailed knowledge proved very valuable, as operations could be maintained without any capital expenditure and without robbing the mine of ore.

## Topographic Setting

The Paracale district has been exposed to erosion since late Tertiary time and has acquired a fairly mature topography, which to a large extent reflects the character of the underlying rocks. Granite, diorite and the locally indurated sediments usually form rounded hills, rarely exceeding 1000 ft. in elevation. Plains and low areas are often underlain by less indurated sediments.

The sea is transgressing at present, as weathering reaches some distance below sea level in places and the coast line is of the drowned type, with tide water reaching far inland in sluggish rivers, flowing through broad, mangrove-studded valleys.

Some 10 to 15 miles inland is a recent volcanic belt, which can be traced across several provinces to the southeast, characterized by very young topography and rugged volcanic peaks up to 8000 ft. in elevation. The volcano Mayon, in the province of Albay, is still active.

The Larap deposits occupy one of several ridges extending out into the Pacific Ocean. The ridge is divided into three hills, of which the northernmost is the island of Calambayungan. A 2000-ft. wide sound separates it from the Larap peninsula, with its two roughly equal hills on the Bussar and the Bessemer-Rescue claims, respectively. The intervening saddle is on the Superior claim. These hills are topographic expressions of the ore-bearing structure and its hard hematite outcrops. The sound and the saddle are probably expressions of east-west faults, as shown in Fig 1.

Farther south the ore-bearing formation swings to the east and continues on the next peninsula to Dawahan, where hard, dense metamorphics form a line of hills up to 500 ft. elevation, dissected by fault-controlled cross valleys.

This part of Luzon has no definite dry season. A rainy season occurs from October to March, while the northeast monsoon prevails. Typhoons occur in October, November and December. Some years may be entirely free from typhoons; in others, there may be several—for example, in 1937 there were seven, of which two were of destructive intensity. Exceptionally good drainage must be provided to handle typhoon rainfalls of 24 in. per diem in the open pits, particularly where the workings are in the soft, lateritic surface material. During the three months mentioned, the management was in steady communication with the Weather Bureau in Manila, so that typhoon damage could be reduced to a minimum by preparing for the storms.

## Geology

The Paracale district is on the flank of a geosyncline, with southerly or southwesterly dips prevailing. Small islands offshore show outcrops of crystalline schists of probably early Tertiary age. Overlying these, and probably to some extent intruded in them, are basic intrusives, mainly peridotite and its metamorphosed phases.

Unconformably overlying the basic intrusives is a series of Miocene sediments, largely argillaceous in character, but with

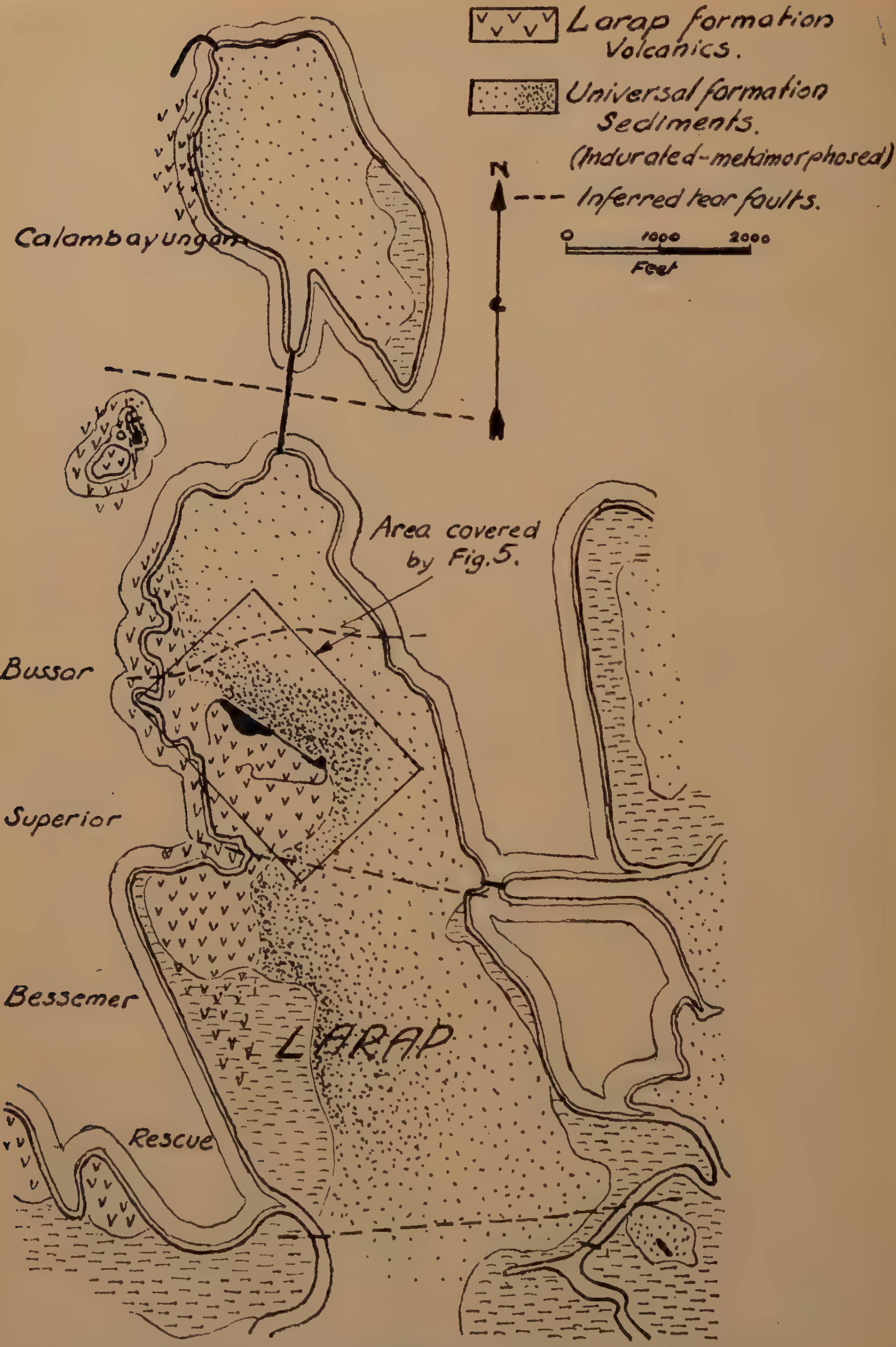

FIG. 1.—PLAN OF LARAP PENINSULA AND CALAMBAYUNGAN ISLAND.

subordinate arkose, quartzite, limestone, conglomerate and bituminous beds as well. From its type locality, it has been called the Universal formation. With certain exceptions, referred to later, it is fairly fresh or slightly indurated.

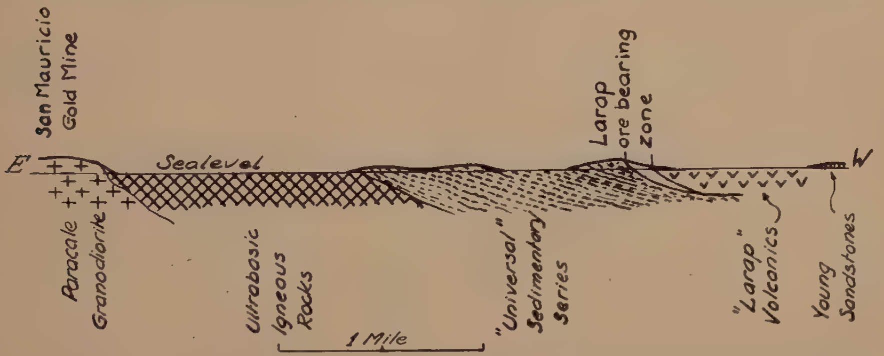

FIG. 2.—EAST-WEST PROFILE FROM SAN MAURICIO GOLD MINE TO BEYOND LARAP.

Intruded in a dome-shaped structure, and with the peridotite as hanging wall, is the Paracale granodiorite, the host rock of important gold deposits. This body extends in an east-west direction for about 9 miles, and is 3 miles wide.[4]

Larap is at the western end of the district. The Universal formation here strikes somewhat east of south. It gradually changes to a more easterly strike around the south side of the dome and across the lower half of the district. Areal distribution of the peridotite rather than actual field observations suggests that the strike then resumes a southeast direction. The district thus represents a somewhat S-shaped, monoclinal structure with some closure around the granodiorite dome on its north flank.

One to several miles from the granodiorite contact, the Universal formation contains basalts and andesite lavas and associated fragmental volcanics. In this zone are many intrusions of andesites and diorites, which have metamorphosed the sediments in an intricate manner to hornfels, porphyroblastic rocks and occasionally coarser, granoblastic rocks. The metamorphism has been a thermal one, as no schists are found.

In this zone new metallic material has been added both hydrothermally and metasomatically, particularly magnetite, with pyrite, pyrrhotite, chalcopyrite, molybdenite and gold in order of decreasing quantity.

The only commercial deposits are those of magnetite. Besides Larap, another iron deposit has been opened at the eastern end of the district, south of the town of Paracale. Between the two, several smaller ones have been explored and to some extent mined.

It is natural to connect the iron deposits with the diorite intrusions, and probably there is such a genetic relationship between the two, although in Larap the ore is clearly related to syenite porphyry, which may have been derived from a diorite magma.

Fig. 2 is an east-west profile from the granodiorite contact to Larap and beyond. The ore-bearing zone occurs between the Universal formation and the volcanics, in this case represented by unconformably overlying agglomerates, which cover a considerable area west and south of Larap and therefore have been given the name of Larap formation. Above this are in places undisturbed young sandstones, of no significance in this connection.

A crescent-shaped wedge of the contact

between the Universal and the Larap formations is metamorphosed and mineralized in a zone of varying width and extending for at least 6 miles in a north-south direction. The following descriptions refer to this zone, a typical ore-bearing section of which is shown on Fig. 3, and include shale, argillite, metamorphic sediments and agglomerates of the original rock section. Introduced material and rocks are syenite porphyry, skarn rock, andesite dikes, magnetite-pyrite ores, massive magnetite ores and other mineralizations.

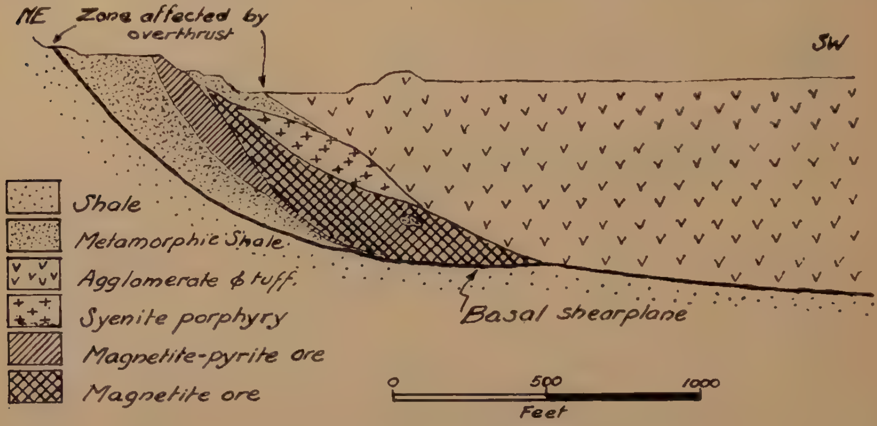

Fig. 3.—Section through ore-bearing zone, Bussar claim.

### *Shale*

Near the ore zone the bulk of the Universal formation is made up of gray, black, brown and green shales and graywackes, types with poorly developed stratification dominating. Beds of sandstone and conglomerates have been observed, as well as a thin bituminous seam, and in one locality a brown, slightly manganiferous bed, the decomposition of which left a few small pockets and boulders of manganese ore of good grade.

Table 2.—*Analyses of Rocks and Ores, Bussar Claim*
Per Cent

| Constituent | Shale | Metamorphic Sediments | Agglomerate, Larap Formation | Syenite Porphyry | Light Portion of Migmatite, Partial Analysis | Dark Portion of Migmatite, Partial Analysis | Skarn Rock | High-grade Magnetite-pyrite Ore, Partial Analysis | Low-grade Massive Magnetite Ore, Partial Analysis |
|---|---|---|---|---|---|---|---|---|---|
| $SiO_2$ | 52.12 | 48.43 | 52.64 | 56.60 | 63.77 | 48.55 | 48.18 | 7.46 | 9.17 |
| $Al_2O_3$ | 20.99 | 17.20 | 17.78 | 17.55 | | | 3.83 | | |
| $Fe_2O_3$ | 5.49 | 9.04 | 7.08 | 7.37 | 4.01 | 13.21 | 6.88 | | |
| FeO | 3.73 | 3.07 | 2.37 | 0.79 | 0.22 | 2.33 | 4.91 | | |
| MgO | 1.70 | 2.37 | 1.58 | 0.66 | 0.29 | 1.34 | 6.05 | 0.04 | 0.31 |
| CaO | 3.60 | 2.22 | 5.71 | 2.84 | 1.11 | 5.61 | 21.82 | 0.26 | 2.16 |
| $Na_2O$ | 3.75 | 7.90 | 7.23 | 4.59 | 3.72 | 1.86 | 4.21 | | |
| $K_2O$ | 2.42 | 3.95 | 4.12 | 8.59 | 2.11 | 1.05 | 1.69 | | |
| $H_2O$ | 6.26 | 3.97 | 0.91 | 0.44 | | | 1.49 | | |
| $P_2O_5$ | 0.29 | 0.20 | 0.59 | 0.59 | | | 0.38 | | |
| Fe | | | | | | | | 58.49 | 57.80 |
| S | 1.35 | 0.72 | 0.20 | | | | 0.17 | 5.92 | 0.48 |
| P | | | | | | | | 0.16 | 0.05 |
| Total | 101.70 | 99.07 | 100.21 | 100.02 | | | 99.61 | | |

Several samples of the shale have been

examined under the microscope. It is a fine-grained rock in which can be identified feldspar, quartz, epidote, calcite, chlorite and sericite, with the feldspars dominating. In one sample a specimen of *Globigerina* was found. A chemical analysis is given in Table 2. The shale is a subsiliceous rock with fairly high alumina content.

### *Argillite*

The metamorphism of the shale begins as a thin band of argillite. It is a hard, dense rock with conchoidal fracture and of light color in grayish and greenish shades. The often bright colors of the shale are thus largely lost in the induration. Great as the difference is between shale and argillite in hand specimens, it is barely possible to distinguish the two under the microscope, as there is as yet no appreciable increase of grain size. The argillite on the Bussar claim is often brecciated and cemented with pyrite.

Where the actual contact between the two rocks is not of tectonic origin, the induration has proceeded along irregular, wavy planes, which have a tendency to be convex toward the shale. The actual contact is sharp and resembles strikingly the sharp front of tropical weathering, caused by descending acid and oxygen-charged surface waters. It appears that the induration of the shale to argillite was caused by the progress of juvenile, warm waters, the vanguard of the action, which in the rear deposited the ore.

### *Metamorphic Sediments*

Metamorphic sediments form the host rock of the magnetite-pyrite ore and the footwall of the massive magnetite ore; and locally also the hanging wall.

This rock, in its typical development, is a hornfels with an interlocking mosaic of feldspar and other minerals. It is a more intensely recrystallized phase of the shale than the argillite, with an appreciable augmentation of grain size. Variations in the composition of the original shale, different intensity of metamorphism and metasomatic changes have given rise to a multitude of different rock types and gradations, of which a few are referred to below.

On the Bussar claim the rock consists mainly of plagioclase and orthoclase with smaller amounts of biotite, quartz, hornblende, chlorite and calcite. Magnetite is always present either as dust throughout the rock or as characteristic "footprints" of scattered groups of small magnetite grains, often surrounding small feldspar aggregates. Idiomorphic crystals of magnetite are rare.

The appearance of the magnetite indicates that it was introduced during the recrystallization. The analysis in Table 2 and the drill-hole analyses of Fig. 4 show that much magnetite was introduced. The same holds true for pyrite, which occurs as somewhat larger, poorly developed skeletons. Calcite and chlorite are later, secondary minerals.

Where the original shale was more calcareous, calcite with small diopside crystals occur as interlocking grains. Occasionally the whole rock is transformed into a calcite-diopside rock, not to be mistaken for the skarn rock described later.

In a general way the metamorphism is intensified toward the upper contact of the metamorphic sediments. Near the contact with the Larap formation are occasional zones of coarser, granoblastic rocks, in which magnetite has finally succeeded in crystallizing in idiomorphic crystals with the general enlargement of the rest of the minerals. The most intensely metamorphosed rock is a coarse cordierite gneiss occasionally observed in drill holes near the contact, made up of a mosaic of cordierite, biotite, plagioclase and magnetite.

The zone of metamorphic rocks on the Bussar claim is seldom more than 300 ft.

thick, the rest of the space between the shale and the Larap formation being occupied by ore and syenite porphyry. With the typical crescent shape, it tapers off down the dip and disappears, so that the overlying Larap formation rests directly on shale or argillite.

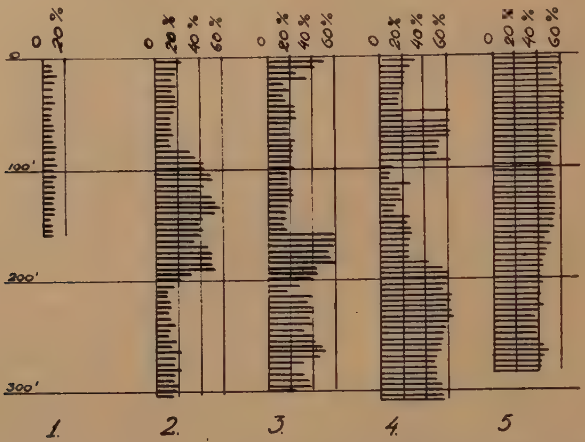

FIG. 4.—IRON CONTENT OF 5-FOOT SAMPLES IN CHURN-DRILL HOLES.

To the north on Calambayungan the zone is much thinner, probably not more than 100 ft. thick, and the dip is flatter, all indicating that the surface has intersected the ore zone nearer its bottom. On the Bessemer-Rescue claims it is much thicker, drill holes indicating some 1000 ft. of metamorphic rocks. A hole 800 ft. deep, starting near the upper contact, penetrated rocks of decreasing metamorphism and was bottomed in argillite, interbedded with hornfels, in contrast with the definite contact between the two in Bussar.

Characteristic in this section is a "bead" texture. The light colored beads, which occur like strings of pearls (and about the same size) in a gray, finely mottled matrix, are fine-grained aggregates of orthoclase with but few dark minerals. The matrix contains in dustlike form dark minerals, mostly magnetite. With the low crystallization force of the orthoclase, it is assumed that the bead texture was caused by introduction of magnetite into the pores of the rock after the feldspar had begun to recrystallize along planes of stratification, whereupon the process was arrested.

Approaching the hanging wall, coarser granoblastic rocks of orthoclase and biotite become common. One type is a light green, tough orthoclase rock, which in hand specimens reflects light simultaneously over areas of square inches. In the microscope it is a medium to coarse-grained orthoclase with sericite. With crossed nicols the sericite is shown to be oriented parallel over large areas, regardless of grain boundaries, which obviously is the cause of the light reflection phenomenon. No other explanation is offered than that it is a strain phenomenon.

The most intense metamorphism in the Bessemer-Rescue area consists of zones of large, pink orthoclase with masses of biotite, muscovite, radial columns of amphibole and neutral colored prisms of apatite, to some extent mineralized with chalcopyrite, pyrite and molybdenite. Many intermediate types prove it to be a further recrystallization of the granoblastic orthoclase-biotite rock together with introduction of new material.

In the Dawahan area lack of mine workings in unaltered rock has limited mapping to a few drill holes, outcrops and float. The rocks, of clearly sedimentary origin, are similar to those in Bussar except that the ore minerals include pyrrhotite and the rock minerals epidote, both indicating a higher prevailing temperature. Some garnet-amphibole rocks occur.

### *Larap Formation*

The dominant rock in the Larap formation is an agglomerate, but some tuffaceous material has been observed. The most typical development is along the western shore of the Larap peninsula and on small islands offshore, where wave action has brought out its character. It is a fragmental rock with an abundance of subangular fragments of latite. Occasionally larger, rounded boulders are found. The matrix is arkosic. In the Dawahan area the arkosic material dominates, so that often it is difficult to recognize the rock, particularly when some metamorphism has obliterated its fragmental character. An analysis of the Bussar rock is given in Table 2.

The typical rock forming the hanging wall of the ore zone is gray, very hard, and massive, and in it the subangular fragments are faintly outlined against a somewhat darker, mottled matrix. The microscope shows it as an irregular, granular to fine-grained mosaic of plagioclase, orthoclase, augite, epidote, biotite, hornblende and albite, with the feldspars first mentioned dominating and in part forming phenocrysts. Accessory minerals are apatite, zircon, pyrite, magnetite and titanite, with chlorite, sericite, leucoxene and calcite as secondary minerals. Sutured grain boundaries show that the rock is metamorphosed. A more intense metamorphism tends to blend the fragment and matrix so that spotty appearance is all that remains of the fragmental structure.

The outstanding difference between the Larap formation and the underlying Universal formation is that the former as a whole is a competent rock and the latter is incompetent except where metamorphism has imparted some competency locally.

The Larap formation, as mapped and drilled (compare Fig. 3), may well have been deposited in a valley, eroded in the Universal formation. This would account for the flat floor and the sudden rise of the contact in a manner strikingly resembling the floor and side of a river valley approaching maturity.

### *Syenite Porphyry and Migmatite*

Part of the contact between the ore-bearing zone and the Larap formation is occupied by syenite porphyry and associated rocks. In this position it is exposed on the Superior claim. A similar body is known in considerable detail from many drill holes on the Bussar claim, where it sometimes is separated from the overlying agglomerates by iron ore or metamorphic sediments. Numerous fragments of float have been shed from a similar body in Dawahan.

In its typical development it is a pink, porphyritic rock with medium sized, somewhat cloudy orthoclase phenocrysts in a fine-grained matrix, which often has a greenish tint due to chlorite and other minerals. The matrix consists of orthoclase and plagioclase, nearing oligoclase in composition. Some biotite, pyroxene and hornblende occur in minor quantities. Chlorite is common, quartz is rare. An analysis of the rock is included in Table 2.

The complete story of the remarkable porphyry body on the Bussar claim will not be known until it has been exposed by mining operations. In section roughly crescent shaped (Fig. 3), it forms a lens on or just below the agglomerate contact (Fig. 6), and has to some extent been displaced by faulting, to which reference will be made later.

In the overlying agglomerates the porphyry has caused considerable metasomatic changes, giving rise to hybrid rocks—migmatites. These occur even where iron ore separates the porphyry from the agglomerates, indicating that the ore in these places is younger than the migmatites.

Two clearly distinguished types of migmatite are particularly interesting. In the first type, and presumably the older one, the agglomerate structure is still preserved. The usually gray matrix has received additional dark minerals—pyroxene, hornblende, epidote and magnetite. Later the fragments themselves begin to be attacked along the borders. At this stage the rock looks like a conglomerate, with light, diffusely outlined "pebbles" in a dark matrix. An analysis of the dark matrix (Table 2) shows an increase of iron, calcium and magnesium.

In the second type the recently deposited dark minerals are swept away into peripheral areas, leaving a white matrix, which blends with the fragments so that they can hardly be distinguished. This process took place along many fronts in a layer a fraction to a couple of inches wide. At one stage the resultant rock stands in a baffling contrast to the first type, particularly when, as often happens, they are seen together. Remnants of the former dark matrix remain as diffusely outlined *dark* "pebbles" in a *white* matrix!

An analysis (Table 2) confirms the guess that this rock has lost most of its iron, calcium and magnesium and has regained lost alkalies, and that it surpasses both the syenite porphyry and the agglomerates in silica. In composition it stands near granodiorite and somewhat resembles aplite in hand specimens. In the microscope it exhibits typical metamorphic textures with interlocking grains and general recrystallization. Some phenocrysts are still recognizable.

While the syenite porphyry was intruded, one may imagine the rock heated up rapidly and volatile or soluble matter, particularly iron, calcium and magnesium compounds, was deposited metasomatically. With a further increase of temperature to near fusion point the dark minerals were brought back in solution and redeposited in peripheral, cooler areas. The remaining rock received new material, particularly silica, from the magma, and after cooling formed a hybrid rock, in which the elements were part of the agglomerate and part of the syenite magma.

The point to remember is that the temperature near the contact rose rapidly to near fusion point and subsided equally rapidly, as otherwise the phenomena described would not have had such local character as to be practically telescoped into each other. Furthermore, the syenite porphyry intrusion was metasomatically potent, in contrast with ordinary dikes. In part this may well be due to the tightly closed space into which the porphyry was intruded, a true "chonolith," which permitted no easy escape in any direction for solutions or gases under high pressure.

### *Pyroxene Skarn*

Skarn includes various coarse, metasomatic rocks, usually made up of aggregates of lime-magnesia-iron silicates and other minerals like quartz and biotite. Such rocks may be seen on the contact between limestone and igneous rocks, for example, but they are most characteristically found together with iron or sulphide ores of metasomatic or hydrothermal origin.

A typical skarn is found in the Bussar area as an irregular zone near and beyond the eastern end of the porphyry body. Usually confined to the contact with the Larap formation, it also occurs inside the metamorphic sediments. Iron ore may form one of the walls of the zone.

It is a coarse rock, the bulk of which consists of columnar aggregates of dark green pyroxene (hedenbergite) with actinolite, muscovite, biotite, epidote, some

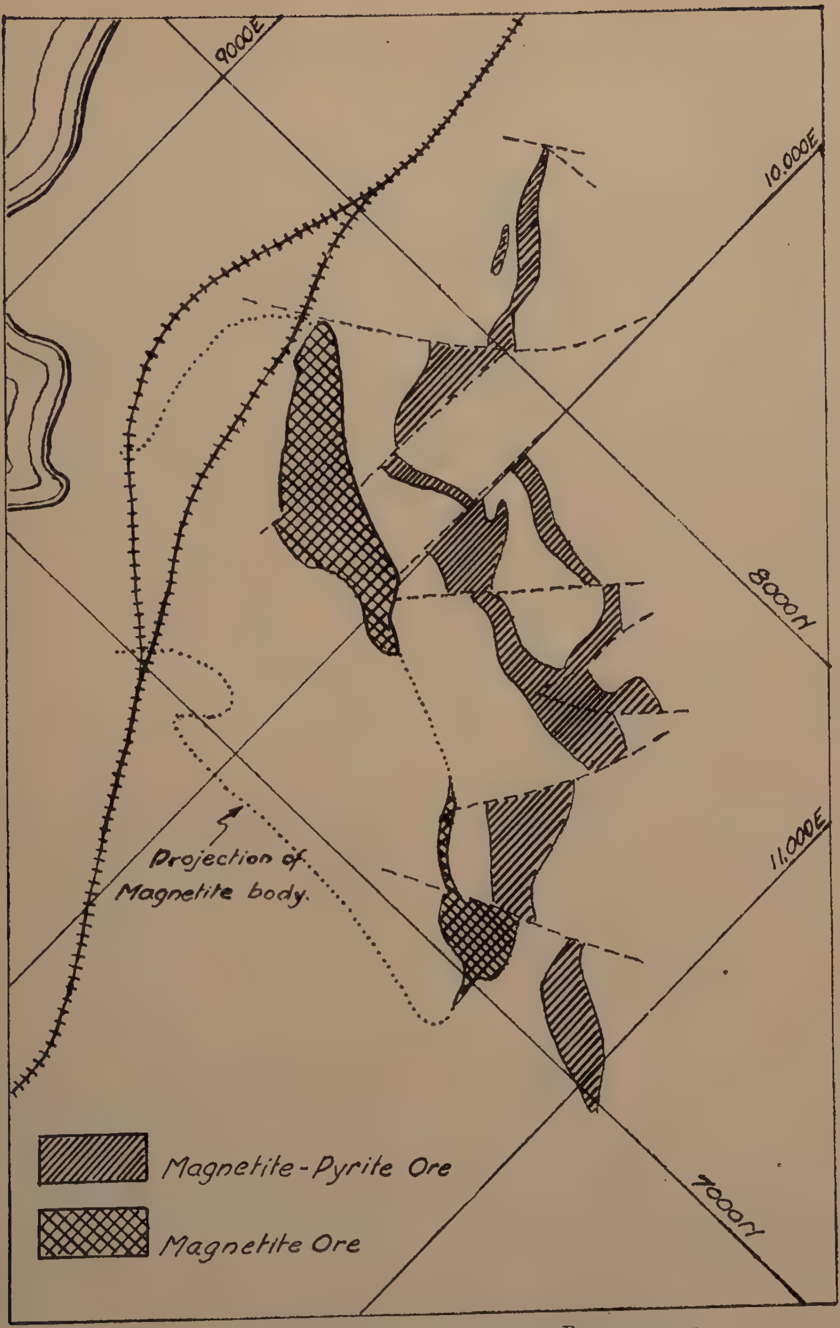

FIG. 5.—SURFACE PLAN OF ORE BODIES ON BUSSAR CLAIM.

quartz, and calcite and apatite, of which the two last named appear to be somewhat later than the others. The apatite occurs as large crystals, some as much as an inch in diameter.

The analysis in Table 2 shows a rock that has little resemblance to any original rock in the area, an almost complete replacement by the lime-magnesia-iron silicates, which form some 90 per cent of the rock.

The contacts are gradual. Pyroxene has been introduced in small crystals in the rock, which changes first to a fine-grained, light green rock with silky luster. Within a few feet the typical, coarse skarn rock has developed.

Occasionally skarn is in contact with the magnetite ore. The drill cores never show skarn and ore in the same piece of core; invariably there are two adjacent pieces—one ore and the other skarn. It is evident that the contact is sharp and that there is a parting along it, but as the contact has actually never been seen it is premature to draw any conclusions as to time relationship. The author believes that the skarn is later, although it did not attack the ore.

Field relations strongly suggest that the skarn is intimately connected with the syenite porphyry intrusion, whose volatile phases penetrated deeper into the fault-controlled channels than the viscous magma and caused widespread replacement along these channels.

As the only observed source of these metasomatic changes is the widespread syenite porphyry, it is justifiable to include the migmatites, the skarn, the cordierite gneiss, the mineralized orthoclase-mica-amphibole rock in Bessemer-Rescue and the various epidote-garnet-amphibole-pyroxene-cordierite rocks in Dawahan among the results of metasomatic action of the porphyry, or its source magma, under different time-space-temperature conditions, modified by the chemical character of the replaced rock.

### *Andesite*

In several places, from Calambayungan to Dawahan, are dikes of andesite, not unlike those mentioned earlier in the geologic setting of the district. One of these occurred in the main pit on the Bussar claim—a purple, brecciated rock with numerous plagioclase phenocrysts in an aphanitic matrix. In a specimen of this rock, much of the matrix had been replaced by magnetite, which had left the phenocrysts untouched.

The andesite is older than the ore formation and probably is a dike phase of the Larap formation of agglomerates.

### *Mineralization*

In unaltered condition the rocks in Larap contain only small quantities of metallic minerals. Drill holes in the shale persistently give 8 to 9 per cent Fe. The agglomerates also carry around 6 per cent Fe.

On Fig. 4 is plotted the iron content in 5-ft. sections of some churn-drill holes selected to show types of mineralization. The first hole is in shale, the others are drilled in the zone of metamorphic sediments and obviously have received large amounts of iron.

Parts of holes 2, 3 and 4 carry a steady iron content of about 20 per cent in magnetite and some pyrite. This is the "dust" and the "footprints" of magnetite and the occasional pyrite skeletons mentioned previously, and the evidence is that this roughly corresponds to filling of pore space in the original shale. This type of mineralization has no economic significance. Only the higher grade sections, from the narrow zones in hole 3 to the 300-ft. section of 40 to 65 per cent ore in hole 5, are of interest.

These high-grade ores can be separated into two sharply divided classes. One type contains a harmful sulphur content of 2 to 6 per cent, and is referred to as magnetite-pyrite ore; the other is a massive, higher grade magnetite, with usually less than 0.5 per cent S and a syngenetic lime-magnesia silicate gangue, and is called massive magnetite.

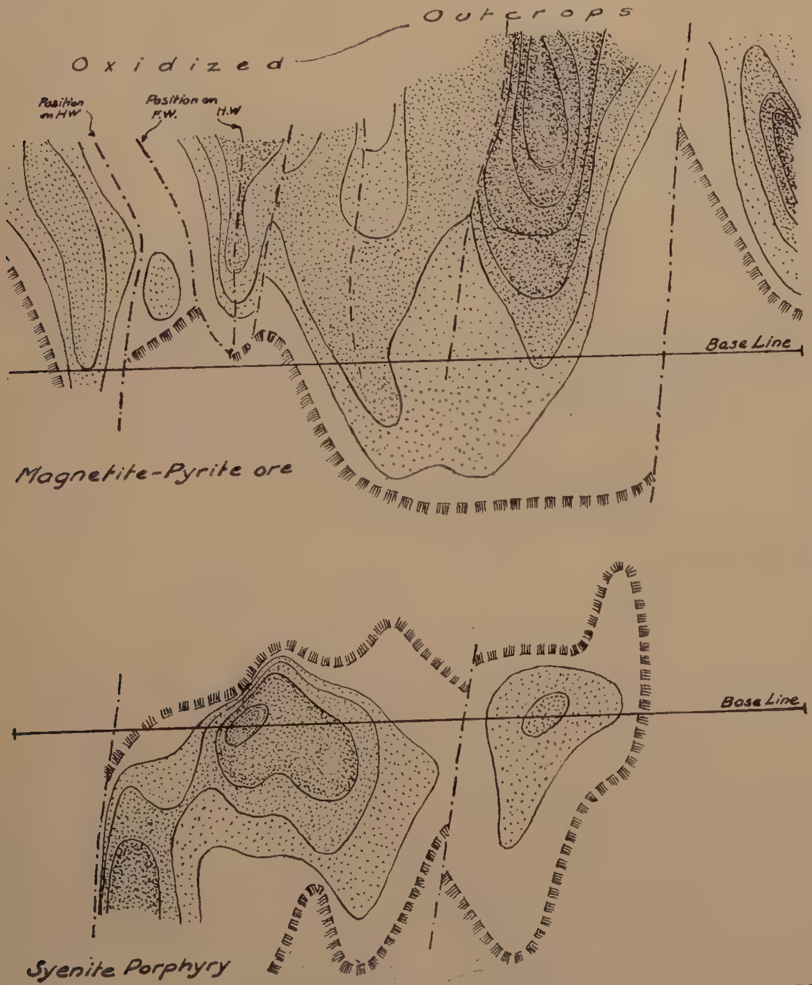

FIG. 6.—PROJECTION ON A PLANE DIPPING 45° SW. OF MAGNETITE-PYRITE ORE, SYENITE PORPHYRY AND MASSIVE MAGNETITE.

Except where otherwise stated, the following discussion pertains to the ores on the Bussar claim. These are well known, through the mining of some 3 million tons and from numerous drill holes. On Calambayungan the exhausted ore reserves were all in the oxidized zone; on the other claims mining has been largely confined to the same superficial zone of oxidation and ore reserves are only roughly outlined by drilling.

### *Magnetite-pyrite Ore*

As shown on Fig. 4, few analyses fall between 20 and 40 per cent Fe. Those above 40 to 45 per cent gather into regular ore bodies, which have or will have economic significance. This ore contains magnetite and pyrite in a ratio varying between 85:15 and 95:5. The pyrite does not contain appreciable amounts of copper or gold. Gangue minerals are those of the metamorphic sediments. Magnetite and pyrite thus were the only minerals introduced.

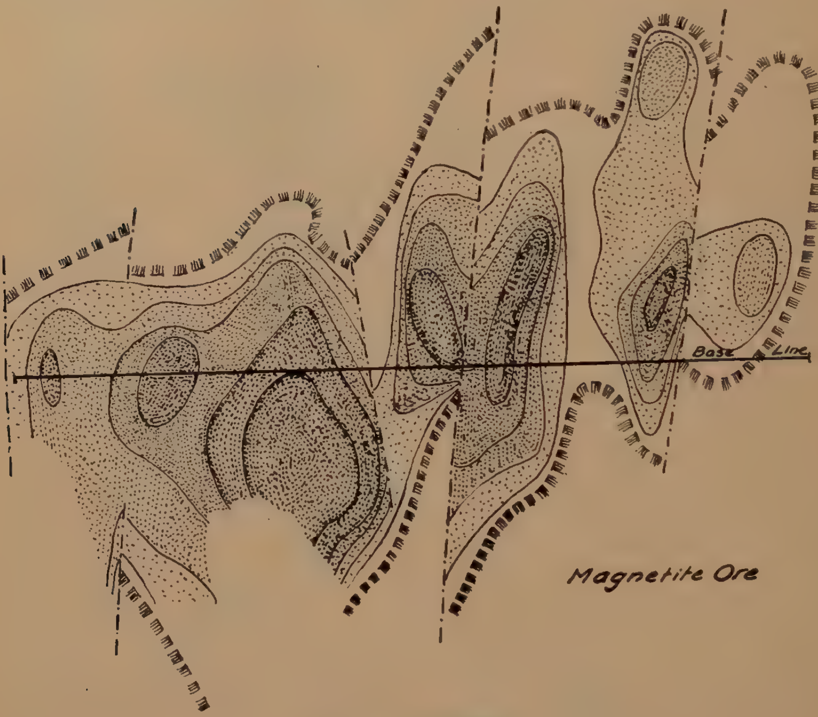

FIG. 6.—(*Continued*).

The texture of the ore varies within wide limits. Coarser grained, high-grade types are often vuggy, with magnetite and pyrite crystals lining the vugs. These types are frequently brecciated and the breccia may be cemented by magnetite or it may be left as a slightly consolidated mass, falling apart during mining.

Finer grained ores are usually of lower grade than the coarse ore. Vugs do not occur and brecciation is not prominent, because the ore is harder, owing to the strengthening effect of the feldspathic gangue. When mining began this ore formed bold outcrops and cliffs of nearly pure hematite. Leaching and oxidation had removed most of the gangue and the sulphur and left a hard, porous ore, in which was preserved the texture of stratification in the original sediments.

Ore boundaries as a rule are sharp, and often slickensided, and within an ore body may occur coarse as well as fine-grained types, high-grade as well as low-grade ores. Among smaller ore shoots are stockworks, fissure filling and breccia cementing as well as indefinite masses of ore without apparent structural control.

One cannot escape the conclusion that these ore bodies must be credited to more than one mode of deposition. The lower grade ores, the ores showing stratified texture, and the indefinite masses without structural control, indicate replacement. The vuggy, coarse, high-grade ore, breccia cementation, and related structures, equally forcefully show hydrothermal filling of structural cavities. If the data on hand have been interpreted correctly, the replacement type dominates.

Disregarding smaller ore shoots, the magnetite-pyrite ore forms a series of well-defined bodies, more or less contiguous for some 2000 ft. in a northwest-southeast direction, and with moderate to flat dips to the southwest. Along the strike the ore is chopped up by faults into podlike masses (Fig. 5). In section each pod has a regular, crescent shape with the concave side up, so that the dip flattens at depth (Fig. 3). The thickness of individual pods is as high as 160 to 180 ft. and tapers to a mere shear at depth (see also Fig. 6). As erosion has removed the apex of the ore, the dimensions along the dip of the pods are unknown, but restoration of structure suggests that these may have been considerably more than 1000 feet.

Adjacent pods seldom, if ever, fit together and the faults separating them seem rather to have modified the shape and size of the pods than to have displaced them in the usual manner. Mining began on many outcrops, separated by barren rock. Later, ore appeared between these, and as greater depth was reached the number of prominent faults decreased and the various pods joined to form more continuous ore bodies.

Stratigraphically the magnetite-pyrite ore bodies occupy a position nearer the footwall of the wedge of metamorphic sediments than the overlying massive magnetite.

### *Massive Magnetite*

Massive magnetite stands in sharp contrast to the magnetite-pyrite ore. Excluding some impure ore near the apex of the ore body, the grade is well above 60 per cent Fe. Pyrite is no longer a prominent mineral in the ore. The gangue of lime-magnesia silicates is syngenetic with the ore.

The gangue is hedenbergite, radiating masses of actinolite, a green mica, some quartz and—much later than the ore—chlorite, calcite and apatite. The latter has been found together with a soft, fibrous mineral as large crystals, reaching sizes of 2 in. by 2 ft., in cavities in the ore, and does not materially affect the phosphorus content of the ore. The order of crystallization has been magnetite, magnetite and gangue, gangue, chlorite-calcite-apatite. Among metallic minerals is some pyrite, which is concentrated near the apex of the ore; some chalcopyrite occurring with late actinolite gangue, and molybdenite as small flakes in partings in the ore. Several characteristic textures have been observed.

1. A very high-grade, brecciated ore, in which the fragments have been rolled and rounded off to "pebbles," cemented by magnetite. Along the footwall, particularly near the apex of the ore, many of these "pebbles" were slickensided, glossy fragments of metamorphic sediments of all dimensions, some showing signs of replacement. The previously mentioned andesite has supplied some of these fragments.

2. A coarse to medium-grained ore with vugs and cavities, lined with small crystals of magnetite, green mica and actinolite. Sometimes specularite and hedenbergite are found as well.

3. A finer grained magnetite with dark green schlieren of the gangue minerals. Excepting the quantitatively meager replacement seen, it is obvious that the massive magnetite is largely a filling of a cavity. The mineral assemblage suggests conditions approaching those of hypothermal deposits.

The Bussar ore body is well known from

mining and from drilling. Other ore bodies of the same type are imperfectly known from other parts of the property.

It is some 1500 ft. long and up to 150 ft. thick. In section it has the same typical crescent shape (Fig. 3) as most other features in the ore-bearing zone. Although it forms a continuous body, the wider portions form pods in the same way as the magnetite-pyrite ore. The hanging wall is formed of agglomerates, syenite porphyry or skarn; the footwall, of metamorphic sediments or, locally, the magnetite-pyrite ore. Part of the western end of the porphyry is embraced by the ore.

Clean partings separate the ore from the walls. The hanging wall in particular is sharp and straight. Several small fissures in it do not enter the ore, but, on the contrary, ore material has penetrated into it. In places the footwall follows a zigzag course between two systems of planes, roughly parallel and perpendicular to the strike of the ore. At no place do these enter the ore. Except for one or two faults, which actually displace the ore, even prominent faults in the footwall area do not enter the massive magnetite ore.

Along the strike of the body the ore pinches gradually to the southeast but disappears rather suddenly at the other end, probably against a fault that has been observed near by. It pinches gradually at depth (Fig. 6). In one place a "handle" some 300 ft. wide extends beyond the drilled area. This handle, incidentally, lies parallel and adjacent to a similar handle of the syenite porphyry body and both may represent channels leading toward the source of both bodies.

### *Other Mineralization*

Occasional references have been made to chalcopyrite and molybdenite.

Chalcopyrite is common. In the main pit on the Bussar claim it occurred in scattered, irregular masses of actinolite near the apex of the magnetite ore.

On the Bessemer-Rescue claims pyrite, chalcopyrite and molybdenite occur as a widespread, weak mineralization throughout large volumes of the metamorphic sediments. The content may occasionally rise to 0.5 per cent Cu but usually is much less. Molybdenum was generally recorded as traces.

Nearer the agglomerate contact in the same area, considerable exploration was carried out on one of the previously mentioned contact-metamorphic zones. The mineralization of chalcopyrite and molybdenite followed the coarse orthoclase-mica-amphibole-apatite rock, which had formed in irregular pockets and masses along the zone. Occasional analyses would run as high as 9 per cent Cu and 4 per cent Mo, but there were no ore bodies and the average was not high enough to permit any bulk mining of the zone.

In a similar fashion, but still less promising, chalcopyrite occurs with pyrite and pyrrhotite in the Dawahan area.

Molybdenite, in addition to the occurrences mentioned, has been observed as bunches up to a couple of hundred pounds in weight in the footwall of the ore in Calambayungan, and to some extent in Bussar; as flakes along partings and postmineral slips in Bussar; as the sole metallic mineral in a quartz vein on the Superior claim; as scattered flakes in the metamorphic sediments in Dawahan; and in veinlets, 1 to 2 in. wide, with quartz and pyrite in the unaltered Universal formation to the east of the ore zone. Although much time and work have been spent following up these leads, either grade or quantity has been insufficient to warrant exploitation.

Molybdenite was among the first ore minerals to be deposited in Larap and it also belongs among the latest ones, as in the postmineral slips and the quartz vein. Its temperature range seems equally wide, from the higher temperature occurrences with magnetite ore to the tiny veinlets in unaltered sediments. It is indeed a versatile mineral.

On the Bessemer-Rescue claims some late quartz-apatite veins occur along east-west tear faults. Occasionally a speck of gold appears, although the vein as a whole may not have more than traces of gold. Other quartz veins have been explored by other companies south of Larap, with negative results.

### *Structure*

Probably the most striking feature in Larap geology is the sudden appearance of a shale underlying crystalline, metamorphic rocks. Where observed, the contact is closely related to a fault, which at depth curves toward the overlying agglomerates until these rest directly on the shale or the thin band of argillite. Walls of other rocks and ores within the metamorphic zone curve in a similar manner, giving rise to the crescent shapes so typical of Larap structure. Furthermore, they are often polished by compressive, differential movements between the walls. It is therefore indicated that the various contact faults are *shear planes of an overthrust.* The stress moved the sheetlike, competent Larap formation eastward, pushing a wedge of crumbling shale in front of the steeper sloping eastern contact.

As often occurs, the basal shear plane gradually curved toward the surface, and thus embraced a wedge of shale, which became accessible to circulating water. Further thrust movements persistently affected the shale, until the basal shear plane had developed in such a position that it was able to counterbalance by internal friction further strain from the intermittently forward-moving Larap formation. After this, and after the shale wedge had acquired a certain competency from the indurating effect of the circulating waters, the accumulating stress of the thrust was repeatedly released by a lifting action along breaks in the indurated shale wedge. The last such break and lift appeared along the contact between the Larap formation and the now metamorphic rock. All this might have passed by unnoticed had it not been for the concomitant intrusion of the syenite magma and its volatile-laden liquids.

Each recurring movement in the overthrust began with compression until accumulated stress caused failure and brecciation, sliding the thrust block up along the break, changing compression to tension and open cavities where the curvature was greatest (Fig. 7*a*).

Each time a new shot of liquid from the magma was injected into the tensile areas, depositing its content there. Finally the viscous syenite magma itself reached the area, after which ore deposition came to a gradual end.

### *Faults and Fissures*

The stress from the recurrent overthrust movement set up shearing couples in the wedge. When rupture occurred three fracture systems should develop according to well-known pattern:* (1) shears parallel and perpendicular to the force couple, (2) thrust or tear faults, and (3) tension fractures.

1. Shears perpendicular to the couple have been observed frequently, but as they are compressive they have not assisted ore deposition. Shears parallel with the couple are the shear planes referred to under the last heading. As the force couple changes direction from horizontal at the bottom of the Larap formation to some 45° higher up, the shear planes are curved, and movement along these will inevitably cause open space along the curvature, as the opposite sides of the shear plane will no longer match. These have served as channels and space for ore deposition and their character as shears is therefore mainly preserved above the apex of the

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* See, for example, M. P. Billings.[5]

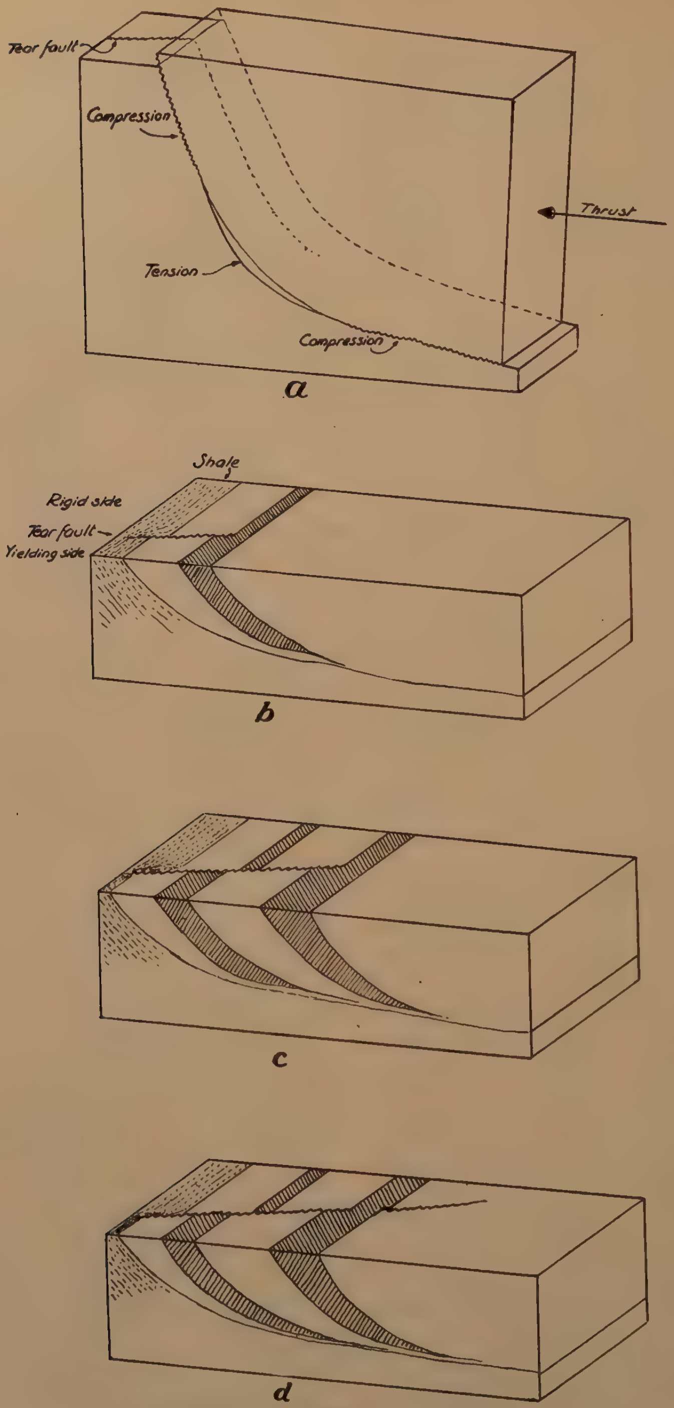

FIG. 7.—TECTONIC RELATIONSHIP BETWEEN ORE AND SHEAR PLANES AND TEAR FAULTS.

ore bodies (Fig. 7*a*), where the stress remains compressive. Thus, above the massive magnetite on the Bussar claim the thrust plane, black and polished, could be followed as a tight fissure to levels above, until it was lost in the oxidized and decomposed surface rocks.

2. Much more conspicuous are the numerous tear faults, which intersect the metamorphic sediments and cut up the ore bodies into the typical pods. A number of these are shown on Fig. 5. As the butts of adjacent pods never match, and as the effect of the tear faults is one of modifying the ore bodies rather than simply displacing them, it is concluded that the tear faults were live faults and played a part in the ore deposition.

The tear faults displace the shale contact and die gradually through strain in the somewhat plastic shale. Upward in the ore zone they penetrate the footwall of the nearest ore body, but may not emerge in the hanging wall. If they do, they may stop in the next ore body. Keeping in mind that the ore deposition took place in and around the tensile parts of the shear planes, the effect of the tear faults on mineralization can be visualized from Fig. 7. As it is extremely unlikely that the metamorphism of the shale produced a homogeneous rock, the tear faults, as well as the shear planes, were localized by the varying strength of the metamorphic rock. Against the oncoming stress, one part would yield more easily than an adjacent part and a tear fault would develop between them (Fig. 7*a*). With repeated thrusting, mineralization deposited more ore on the yielding side of the tear fault, which thus caused some displacement behind the ore without passing through it (Fig. 7*b*). During mineralization along a new shear plane, the same tear fault would again be active, providing more space for ore along the new break and at the same time actually displacing the former ore (Fig. 7*c*). Should the front of the thrust block itself give against the rigid side of the tear fault, the fault will extend into the hanging wall of the thrust zone. The yielding side of the thrust zone will then climb up the shear plane and add more space for ore deposition along the hanging wall (Fig. 7*d*). In this case the tear fault appears on each side of the ore body without passing through it.

Examples of this are shown on Fig. 5 as well as another characteristic feature. Several of the tear faults have here helped to locate the shear planes, in which the magnetite-pyrite ore bodies occur. The shears appear at different places stratigraphically between pairs of tear faults.

Among the structural factors promoting ore deposition, the three most important ones thus seem to have been that: (*a*) the two walls of the shear planes did not match after movements along the planes, thus providing space for ore deposition and channels for replacement; (*b*) repeated thrust stresses, or solution pressure, or both, pushed back the yielding side of the tear faults; and (*c*) the weight of the ore plus the repeated thrust stresses pressed down the yielding side.

In the regional pattern the east-west tear faults dominate the picture. They are present wherever mining or excavation has opened up stretches of fresh rock. East-west valleys intersect the stratigraphically controlled north-south ridges, and where their origins have been ascertained they have always been the same tear faults. Several of these are shown on Fig. 1, but although evidence of faulting is present, the course taken by the faults is inferred from topography.

3. Tension fractures are short horizontal fractures, which obviously must have helped to provide access to the rock for replacement. Of the many tension fractures observed, several carry narrow lenses of ore. From a mining viewpoint, however, they are insignificant and rather a nuisance, as a swarm of them in a drill

hole is likely to deceive the engineering department and exaggerate calculations of ore reserves.

### SALIENT FEATURES OF THE MINERALIZATION

Allowing some latitude in interpretation, a review of the main features of the area contains certain elements worthy of some further consideration:

First, the deposit was formed at a comparatively shallow depth. Had the area been covered by strata measured in many thousands of feet, the thrust stresses probably would have been relieved solely by strain in the shale, but certainly not by a *lifting* action of the front of the thrust block.

Second, the deposition of outside material—ore, skarn, etc.—should not be viewed as static, which would have caused zoning of the introduced material. It should rather be looked upon as a dynamic chain of events from the first stray flows of water of low temperature to the arrival of the syenite magma with its near-fusion temperature of adjacent rocks, and the subsequent return of temperature to normal. Seen in this light the events take on a more interesting character.

There is a zoning within the ore zone—pyrite in the footwall argillites; magnetite pyrite bodies in the center; magnetite, skarn and porphyry along the hanging wall—but field data convey the firm impression that this is mostly due to the chronological sequence of the tectonic events and not so much to areal distribution in a static system

The mineralization can be divided into seven stages.

1. Sulphur and iron were deposited as pyrite from comparatively cool solutions. This mineralization remains in the brecciated argillite.
2. Iron and sulphur were deposited in pores by warmer solutions. This mineralization remains throughout the zone.
3. Iron and sulphur replaced the metamorphic sediments and filled structural cavities from solutions of higher temperature, forming the magnetite-pyrite bodies.
4. Iron, later joined by calcium-magnesium silicates, deposited as the massive magnetite body in a structural cavity. The temperature was high, approaching hypothermal conditions.
5. Calcium-iron-magnesium silicates formed skarn zones and other metasomatic deposits at still higher temperature.
6. Intrusion of syenite porphyry. Some of the previous silicates were moved and replaced by silica in the migmatites. Temperature of fluids reached maximum.
7. Cooling off period. Some massive magnetite probably deposited in structural cavity above the porphyry.

It is remarkable that pyrite practically disappeared from the deposited material between stages 3 and 4. In Dawahan, for example, sulphur was deposited throughout the period of mineralization as pyrite or pyrrhotite, depending upon the temperature. Did the source of sulphur, feeding the Bussar deposits, become exhausted?

It is equally remarkable that in stage 4 most of the ore was deposited in an open cavity rather than as replacement as in stage 3. This in spite of the higher prevailing temperature, and in spite of a return to replacement by the skarn of stage 5.

Did the presence of sulphur in the magnetite-pyrite stage promote replacement of the metamorphic sediments; and, conversely, did the absence of sulphur prevent replacement in the massive magnetite ore? Whatever the correct answer may be, the effect was highly beneficial, as many millions of tons of good iron ore was deposited, high grade because of the mode of deposition, and relatively pure because of the shortage of sulphur.

### PRACTICAL APPLICATIONS

After these general relationships had been established, further exploration and mining became, naturally, much easier. Out of the various indications and geological setups, it was clear which ones should be attacked first, and a look at the cores of a diamond-drill hole was sufficient to establish its location within the ore zone. In a short period, other ore bodies of stage 4 type were discovered and the company's holdings extended to include such ground as would be likely to contain extensions of the vital parts of the ore-bearing structure.

## Acknowledgments

Thanks are due the Board of Directors of the Philippine Iron Mines, Inc., which in 1941 gave the author permission to publish this paper. The author is also indebted to Dean Frasche, Ralph Marsden and Benjamin Meek, formerly of the Bureau of Mines at Manila, who did a great deal of microscopic work on Larap rocks.

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# Geologic Interpretation of Magnetic Exploration on the Mesabi Range, Minnesota

By R. H. B. Jones*

(Chicago Meeting, February 1946)

## Abstract

This paper reviews pertinent geology of the Biwabik iron formation, Mesabi Range, Minnesota. This iron formation is divisible into four members. Each member is distinctive and contains marker horizons, which are recognizable throughout all phases of alteration from lean, fresh rock to high-grade iron ore.

Some beds have a larger magnetite content than others. During the process of ore genesis, which is essentially an oxidation process, the magnetite content is lowered. Alteration from iron formation to ore includes both oxidation of iron minerals and silica removal by leaching and these distinct processes progress at variable rates for different locations and structures.

Magnetic intensities, of each horizon of the iron formation are dependent chiefly upon its preoxidation magnetite content, upon the subsequent degree of oxidation, upon the character of the enclosing rocks, and on the extent of fracturing, folding, or faulting.

The results of a number of magnetic surveys are described and are correlated with results of drill-hole and mine exploration and development. The magnetic anomalies obtained are discussed in relation to phases of formation of ore within the iron formation.

## General Information

One hundred years ago a party of surveyors, applying the principles of geophysical prospecting, discovered iron ore in the state of Michigan in the Lake Superior district. This event was officially reported as follows:[1]

> On Sept. 19 (1844) while running the east line of Town 47 North, Range 27 West, they observed by means of the solar compass remarkable variations in the direction of the needle, amounting to 87° from the normal. Ascribing this phenomenon to iron ore, they sought and found it in ledges or outcrops at several points.

The solar compass referred to was invented by William A. Burt, deputy surveyor in charge of this party.[2] It is used for determining the true meridian by an observation on the sun.

Eight years later, on July 7, 1852, the first iron ore, consisting of 6 bbl. of ore, was shipped from Marquette, Michigan.

It was another 40 years before the first ore shipment left the Mesabi Range in Minnesota, in October 1892. It totaled 4245 tons.

Fifty years later, 90 years after the original shipment, during 1942 the Lake Superior district shipped 93,495,392 gross tons of iron ore. Production came from Minnesota, Michigan, Wisconsin, and Ontario, Canada.

Throughout the 90-year period it was known that magnetic properties of most of the ore bodies, and of the iron formations in which they occur, affected the compass. This phenomenon, later to be made use of to outline areas of ground favorable for exploration, was at first a

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Manuscript received at the office of the Institute Feb. 1, 1945; revised Dec. 13, 1945. The paper was listed for the New York Meeting, February 1945, which was canceled. It was presented at the 18th Annual Meeting of the Minnesota Section, Minneapolis, Jan. 15, 1945. Issued as TP 2038 in Mining Technology, July 1946.

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[1] References are at the end of the paper.

source of real trouble and resulted in some very inaccurate compass surveys.

In order to improve and speed such surveys, Major T. B. Brooks and Prof. R. D. Irving, while working for the Michigan Survey, devised the sun-dial compass.[3] Their dial compass made possible the running of an accurate line and at the same time showed the true direction of the magnetic meridian.

The sun-dial compass and the dip needle soon became standard equipment as an aid in geological field work throughout the Lake Superior district. Hundreds of square miles of territory have been surveyed in this manner by field parties employed by mining companies, state surveys and others. The magnetic surveys have been of great practical value because outcrops in many localities are few or absent and the mantle of glacial drift ranges up to several hundred feet in thickness.

For example, the iron formation of the Cuyuna Range in Minnesota, buried beneath glacial drift, owes its discovery entirely to the presence of strong magnetic variations first noted during a government land survey in 1859. After favorable indications were obtained by dip needle, test pitting was started and ore discovered in 1903.

More recently, during the winter of 1930–1931, a dip-needle survey was run over the ice on Steep Rock Lake in Ontario, Canada. The Steep Rock Lake deposit is not an "iron range" in the true sense of the word, but the survey revived interest in and led to the development of this iron-ore occurrence. Other magnetic surveys as well as electrical and gravity surveys were used at Steep Rock as a check on and follow-up of the dip-needle results.[4,5]

In general, the "iron formations" of the Lake Superior district contain 25 per cent or more iron and are more magnetic than the enclosing rocks. In detail, the ore bodies *within* such iron formations are usually well oxidized and are represented by areas of low magnetic intensities. The contrast between iron formation and ore bodies is determined by the degree of metamorphism, stage of alteration, the location beneath the surface, position within the iron formation and character of the surrounding rocks.

It is evident that often something more sensitive than the ordinary dip needle is required. An answer to this need is the Hotchkiss Superdip, which was developed in the Lake Superior district. This is an excellent instrument for rapid, accurate magnetic work and permits adjustment to give any desired sensitivity. It is used extensively, both for reconnaissance mapping and also for detailed investigations.

Before presenting examples of some Superdip surveys, certain facts about the Biwabik iron formation of the Mesabi Range, Minnesota, are summarized.

## GENERALIZED GEOLOGY OF THE MESABI RANGE

The Biwabik iron formation of the Mesabi Range is of sedimentary origin and is locally called "taconite." It has been traced for a length of more than 100 miles, strikes N.65°E. and dips gently southeastward around 7° to 12°. It is from 500 to 700 ft. thick and is divisible into four distinct horizons named, from top to bottom, the Upper Slaty, Upper Cherty, Lower Slaty and Lower Cherty. Each member contains many marker horizons recognizable in drill choppings, cuttings, core, or outcrop. The taconite is overlain unconformably by the Virginia slate and underlain by the Pokegama quartzite, which is unconformably above slates, greenstone and granite. Minor folding, faulting and one major fold are present in the iron formation, which is truncated by erosion to a feather edge at its northern contact with underlying quartzite. Thin patches of Cretaceous sediments and some older sediments, possibly of Keweenawan age, remain over parts of the range, mostly west of the city of Virginia.

The whole sequence, now exposed in open pits, was formerly largely mantled with glacial drift ranging up to more than 200 ft. in thickness. The taconite beneath the drift has a width of outcrop ranging from about one to three miles, depending on variations of dip, thickness and topography. For all practical purposes, ore bodies are restricted to this width of outcrop.

To generalize: The explorable part of the Biwabik iron formation forms a wedge, north to south, ranging from a few feet to 700 ft. in thickness, over a width of from one to three miles and for a length of more than 100 miles.

The eastern end of the range has been anamorphosed by intrusive Duluth gabbro of Keweenawan age into a dense, hard, recrystallized iron formation containing considerable magnetite and amphibole. Much of the magnetite is considered to be pre-gabbro in age by some investigators.[6,7] This part of the iron formation was anamorphosed before any ore bodies were formed, and it has altered very little since it was recrystallized. The quantity of iron contained in magnetite and other iron minerals in its various members is equivalent to the proportions of iron in the same fresh taconite horizons to the westward.

The effects of intense anamorphism are noticeable about as far west as Mesaba, where the gabbro contact swings southward away from the iron formation.

Katamorphism, or weathering and alteration from taconite to ore, is more complete in the central part of the iron formation. The western end contains "wash ores" in which silica leaching is incomplete and concentration of such ores by milling before shipment has been in progress for more than 35 years.

The quantity of magnetite in the anamorphosed eastern end of the iron formation is not much more than occurs in the taconite to the west. In both places the greater proportion of magnetite occurs in the low-phosphorus cherty members of the iron formation. In fresh to slightly altered cherty taconite, greenalite and siderite zones give way to fine-grained magnetite, which in places is largely restricted to the greenalite granules, but magnetite octahedrons also extend beyond the limits of such granules. This type of alteration, although regional in scope, is confined in large part to the lower cherty and upper cherty horizons and it is not accompanied by any appreciable leaching. It has no relation to structures that have controlled or localized the position of ore bodies.

A second type of alteration in the lower cherty horizon beneath the magnetite zone, and conspicuous enough in many places to form a marker horizon away from ore bodies, is the development of ferric iron bands. In this case, oxidation has been accompanied by some migration of iron. Individual bands, which represent distinct sedimentary layers, in places have joining fractures filled with ferric iron. The oxidation could be pre-gabbro in age and altered to magnetite in the eastern anamorphosed section, but it is probably younger than the gabbro.

## ORE BODIES

The principal processes that formed ore bodies on the Mesabi Range were oxidation of iron minerals and leaching of silica. Most of the commercial ore occurs in trough ore bodies (Fig. 1). There are also fissure and flat-lying deposits. The latter extend outward from trough ore bodies and occur mostly below the base of the lower slaty horizon.

The type of ore resulting from katamorphism varies for different taconite horizons; in other words, the "offspring" ore retains certain characteristics of the

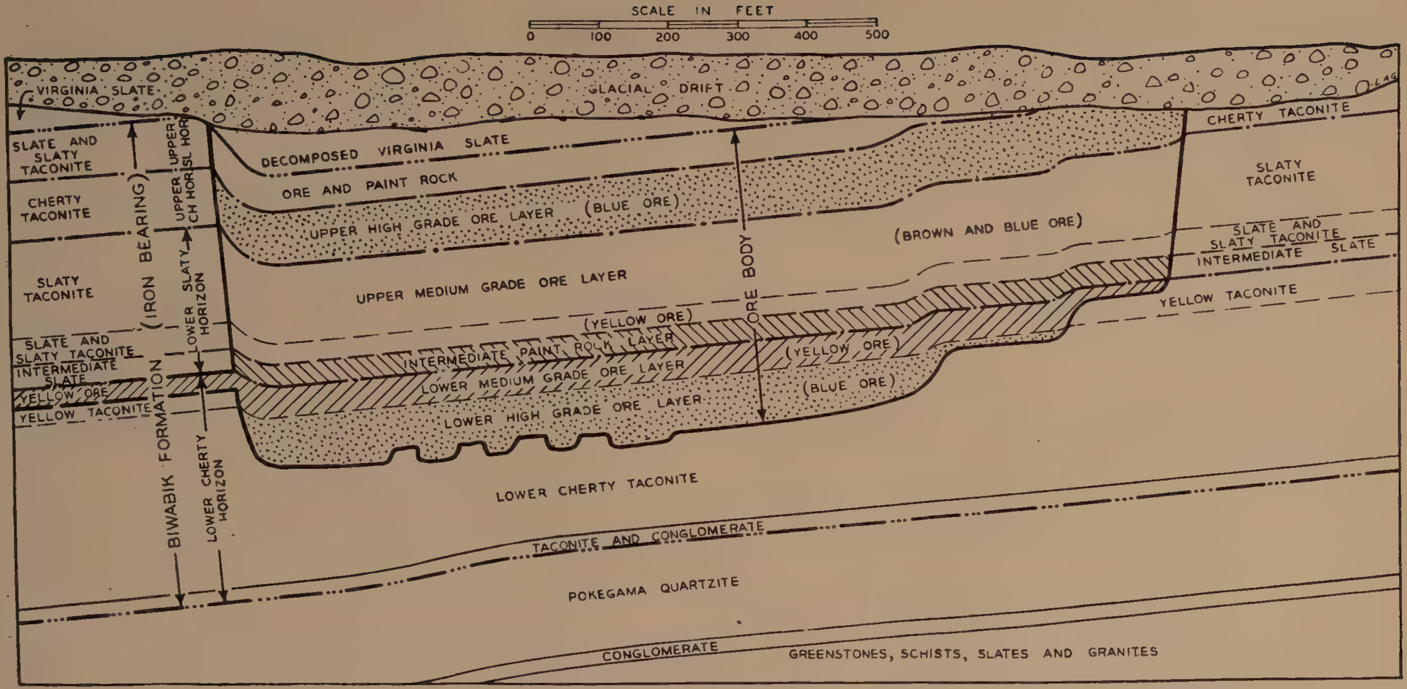

FIG. 1.—CROSS SECTION OF TYPICAL ORE BODY, SHOWING DERIVATION OF ORES FROM BIWABIK IRON FORMATION.

parental bed. The texture of many marker horizons is recognizable throughout all phases of alteration from lean, fresh rock to high-grade iron ore. The grade of ore is also dependent upon the nature of the particular horizon from which it altered, and these characteristics are usually distinctive throughout the length of the range.

Fig. 1 portrays a typical cross section of a Mesabi Range trough ore body and shows the types of ore that are derived from the various horizons. The lower cherty and upper cherty members, in which early alteration produced appreciable quantities of magnetite on a regional scale, when weathered into ore yield a high-grade hematite-martite "blue ore," mostly of bessemer grade.

The yellow taconite that forms the upper part of the lower cherty horizon always alters into a yellowish limonite ore. Alteration products grade from an original greenish greenalite rock with some carbonate through a yellow mottled chert to a yellow limonitic medium grade ore.

The overlying intermediate slate at the base of the lower slaty horizon alters from a black or dark gray slate into a "paint rock" layer; a reddish hematite-limonite high-moisture member of low grade. The "knife-edge" contact between the red paint rock and the underlying yellow limonite is very conspicuous wherever it is exposed. The iron minerals of one layer altered into limonite while those in an adjacent layer weathered to hematite, and there is no evidence of migration of iron from one bed to the other. The "yellow ore" beneath the "paint rock" is never high grade, although it occurs in many ore bodies widely separated along the range, and it is underlain by high-grade ore.

The "yellow taconite" layer weathers more readily than others and the resulting ore usually extends beyond the limits of trough ore bodies forming thin flat-lying deposits, some of which are being mined underground. In contrast, the contact between compact unleached and sometimes unoxidized iron formation and the ore bodies is nearly always a very sharp one, as illustrated in Fig. 1.

This discussion would be unduly lengthened if an attempt were made to describe all the consistent variations of the Mesabi Range taconite. For those interested, a number of references are given at the end of this paper.[8–13] Special reference is made to the detailed work of J. F. Wolff, who worked out and called attention to the various distinctive subdivisions of the iron formation.[11]

*Specific gravity* of the ore is variable. Fig. 1 shows a decided and characteristic slump of the ore layers from their position before alteration from taconite to ore. This slump represents a decrease in volume, which has followed leaching of silica and consequent development of pore space. The resulting slump of such ore may decrease its volume by as much as one third of the size of the original taconite. The porous ore formed from taconite may weigh less than an equivalent volume of unaltered taconite, or with progressive slump it may weigh more. Pore space is always present and the ore retains much moisture.

*This moisture content* of the ore is worthy of note. Slaty layers are always high in moisture content, which may amount to 25 per cent by weight. The ores as shipped average in excess of 10 per cent moisture; hence the very important difference between "dry" and "natural" iron analyses in the iron-ore trade.

With this generalized background, let us proceed to a discussion and interpretation of magnetic exploration.

## FIELD PROCEDURE—HOTCHKISS SUPERDIP SURVEYS

Preliminary to the Superdip magnetic work on the Mesabi Range, experiments were conducted with various instrument sensitivities in order to obtain a practical working range. Greater sensitivities require more exacting correction factors for temperature changes and diurnal variations, and also magnify irregular local attractions. The least sensitivity commensurate with the results desired is the one chosen for use; one that combines both accuracy and speed of operation.

A "sigma" between 3.5° and 4.5° is used, which, on the Mesabi Range, means an angle of about 10.5° to 9.5° between the magnetic axis of the bar magnet and the gravity counterweight arm.[14,15]

This gives a sensitivity about five times greater than can be obtained with an ordinary dip needle. A reading of one degree on the instrument scale is equivalent to about 70 gammas. Zero reading is at the zenith and gradations go to 180° at the bottom of the circle. By comparing readings in a Helmholtz coil and running traverse lines, Superdip and magnetometer surveys were compared and remarkable checks were obtained.

All readings are plotted on a base map showing geologic data. Variations or anomalies and not absolute values are most useful and for convenience the recorded readings may be reduced by some chosen amount.

Readings are contoured on a plan map to simplify the detail and to express the pattern shown by the survey.

In order to obtain legible prints for publication, it has been necessary to eliminate considerable detail in drafting the maps used with this paper.

### *Example* 1

Fig. 2 shows the results of a Superdip survey in an area embracing the northern half of the iron formation outcrop beneath the glacial drift. The upper slaty and upper cherty horizons that outcrop to the south have been eroded from this area. From north to south the thickness of iron formation varies from a few feet to about 350 ft. The open pit to the west was idle for 25 years and it was planned to reopen this deposit.

The ore limits beyond the pit were not fully known. The purpose of the magnetic survey was to outline, if possible, the limits of the known ore body and to determine the ore possibilities of the area between this pit and old stripping dumps to the east. This location could be used for dumps of lean ore and taconite if devoid of iron-ore possibilities.

In general, the known ore body as determined by drilling is fairly well outlined magnetically by the 70° contour and lower readings. The low readings swing eastward from the southern part of the ore body and then trend northward in the shape of a U. These magnetic results indicate oxidation and suggest a possible eastward and then northern sweep of the known ore body. This is a matter of tracing a yardstick of established magnetic readings over a known ore body into an unexplored area with a similar range of readings.

The magnetic pattern to the north consists of a series of elongated north-south-trending "highs" and "lows." These are nearly at right angles to the strike of the iron formation, which in this area is the lower cherty horizon.

If this horizon were unoxidized, the high magnetic contours would parallel the strike of magnetite-rich beds. It was concluded that the lower cherty horizon was oxidized along fracture planes heading into a more uniform oxidation to the southward.

The eastern area of low magnetics and some of the northern "lows" were drilled but no ore was found. Taken at face value, the magnetics might be considered as misleading, so let us refer to the geology.

In the first place, the known ore body to the west is not of merchantable grade. Mining was stopped there 25 years ago.

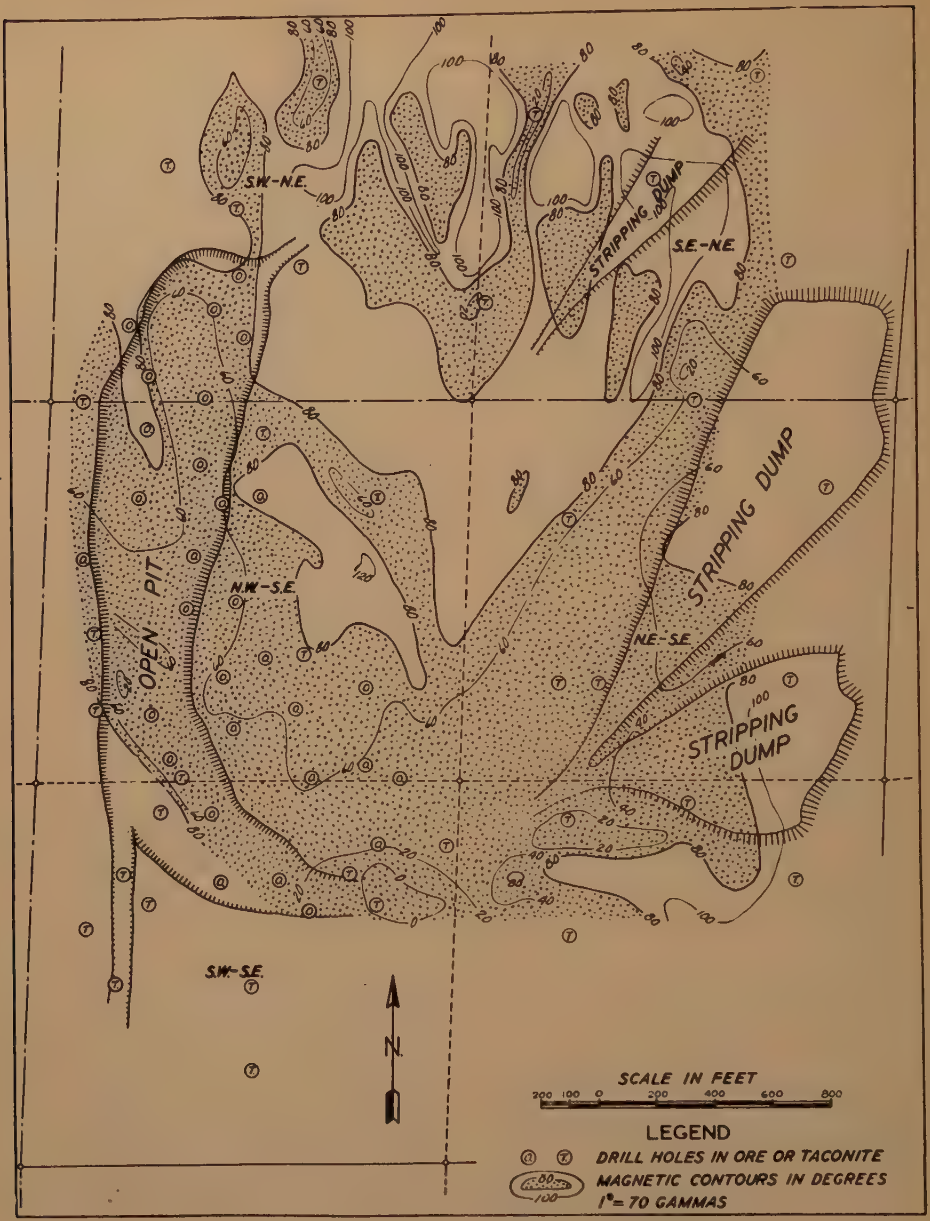

FIG. 2.—RESULTS OF SUPERDIP SURVEY, MESABI RANGE.

The pit is now being operated but much of the ore is concentrated to bring it up to an acceptable grade before shipment. Nature did only part of the job. It oxidized the magnetite, iron silicates, carbonates, etc., but did not leach sufficient silica, much of which remains in a loosened condition, amenable to removal by a simple concentration process. In the area of low magnetics beyond the limit of usable ore today, nature had oxidized the iron, hence the low magnetic readings, but she was less bountiful in the degree of silica leaching. The drill core showed oxidation but very little leaching. Chemical analyses of iron indicated only about average taconite grade.

This is an excellent place to study the first stages of alteration of taconite toward ore. Drilling to the northeast shows preliminary alteration of greenalite with incipient development of magnetite. In the "low" magnetic areas such magnetite gives way to hematite or martite, but with little accompanying leaching of silica. Southward, as a trough structure develops, oxidation is prevalent but silica leaching has not progressed to a point where disintegration permits its removal by wash-ore concentration processes in use today. To the west there is low-grade ore, not of direct shipping grade, but of a type amenable to concentration.

This example illustrates that it is not safe to conclude that low magnetic readings, which indicate oxidation of magnetite, also indicate the presence of commercial ore. Magnetics always have a story to tell but they should not be expected to designate the degree of silica leaching in a Mesabi Range ore body. Parenthetically, the geological implications were known when a drilling program was recommended.

Incidentally other parties interested in the property ran a resistivity survey over part of the area covered by the magnetic work. Their results in general compared favorably with the magnetics, nevertheless they requested additional drilling in proposed lean-ore dump areas where the magnetics were "high." This was one place where the magnetics indicated the absence of any ore. The drilling confirmed this conclusion.

### *Example* 2

Figs. 3*a* and 3*b* show the results of magnetic surveys over the lower cherty horizon along its northern contact with Pokegama quartzite.

The magnetic pattern of Fig. 3*a* is quite erratic, the "lows" are narrow or irregular in shape and bear no relation to "strike" of beds. It was concluded that these low areas indicate localized and spotty oxidation. Taconite core from near-by drilling showed the oxidized and magnetic zones to be very irregular even within narrow limits.

Another item to be considered in the vicinity of Fig. 3*a* is that the glacial overburden is thin, varying from 15 to 30 ft. This is less than average and due to the proximity to magnetic outcrops the readings are jumpy and variable. Each magnetic "high" is mated to a corresponding "low."

The relative regularity of the magnetic pattern of Fig. 3*b* is quite different from that of Fig. 3*a*. In this case the area of low magnetic attraction (minus 80°) parallels the quartzite-taconite contact fairly well. This is probably the magnetic expression of a bed with a low magnetite content rather than an indication of advanced oxidation. For the latter, a less consistent pattern would be expected.

### *Example* 3.—*Selecting a Shaft Location*

A problem often inherited from early operations is the location of shafts, mining locations and even towns on top of ore bodies. At one place a town had to be moved and the former townsite is now part of an open pit. Shaft locations are now selected with care.

Fig. 4 illustrates a property where mine buildings and operating shaft are underlain by ore and a sizable shaft-ore pillar is required to support shaft and surface layout. The complete section of the Biwabik iron formation is present in this mine. An ore horizon in the lower cherty member below the "paint-rock" layer, is mined underground. There is also a wash-ore zone near the surface in the upper cherty horizon and part of it has been mined by open pit.

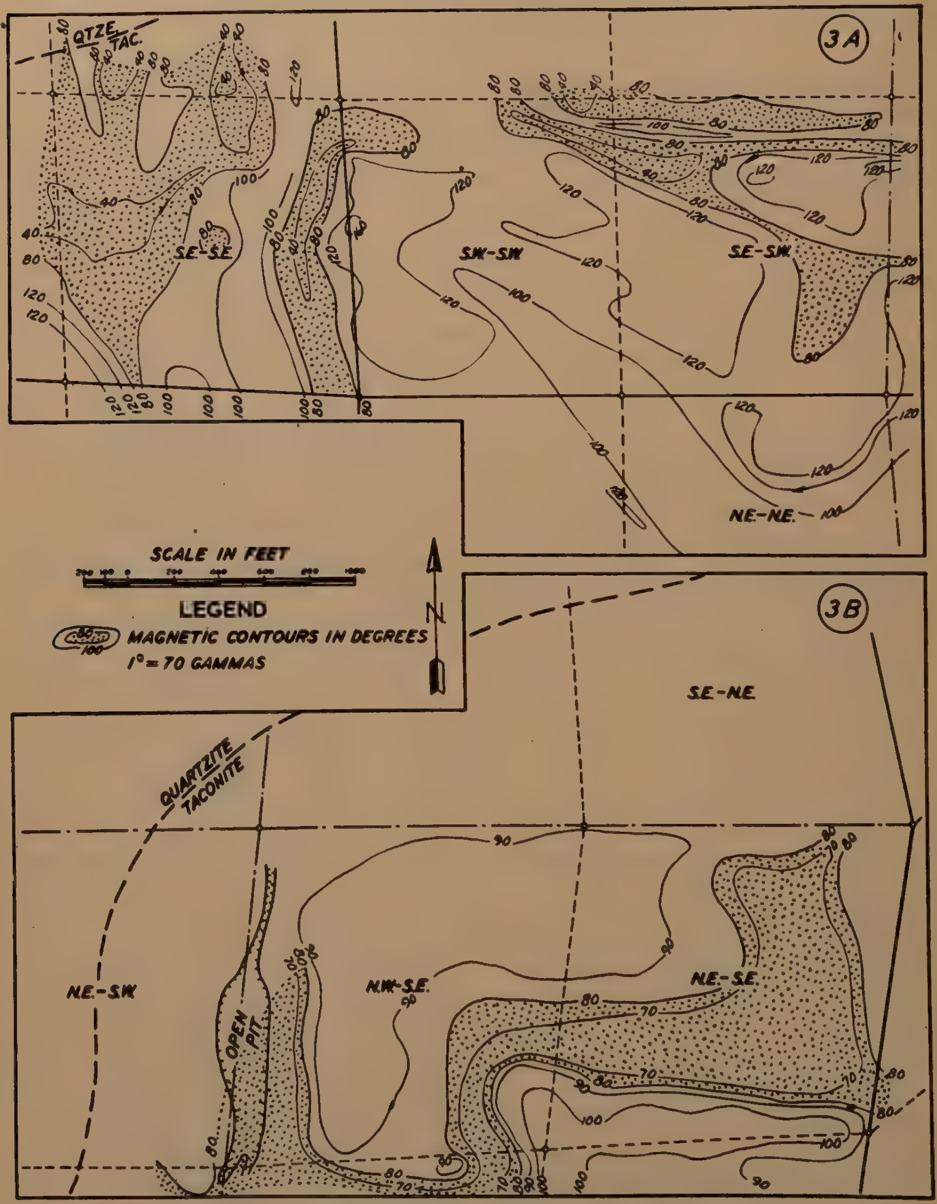

FIG. 3.—RESULTS OF SUPERDIP SURVEYS, MESABI RANGE.

A new shaft was necessary to release the ore tied up by the old shaft. The new shaft location had to satisfy three requirements: (1) it must not be underlain by ore, (2) it must be clear of the future limits of any open-pit operation for the wash ore, (3) it must fit certain restrictions imposed by necessary railway tracks and yard, a large lean ore dump and other cultural features.

The owners of this property made a resistivity survey from the surface and also from underground workings. The operators agreed to check their work with a

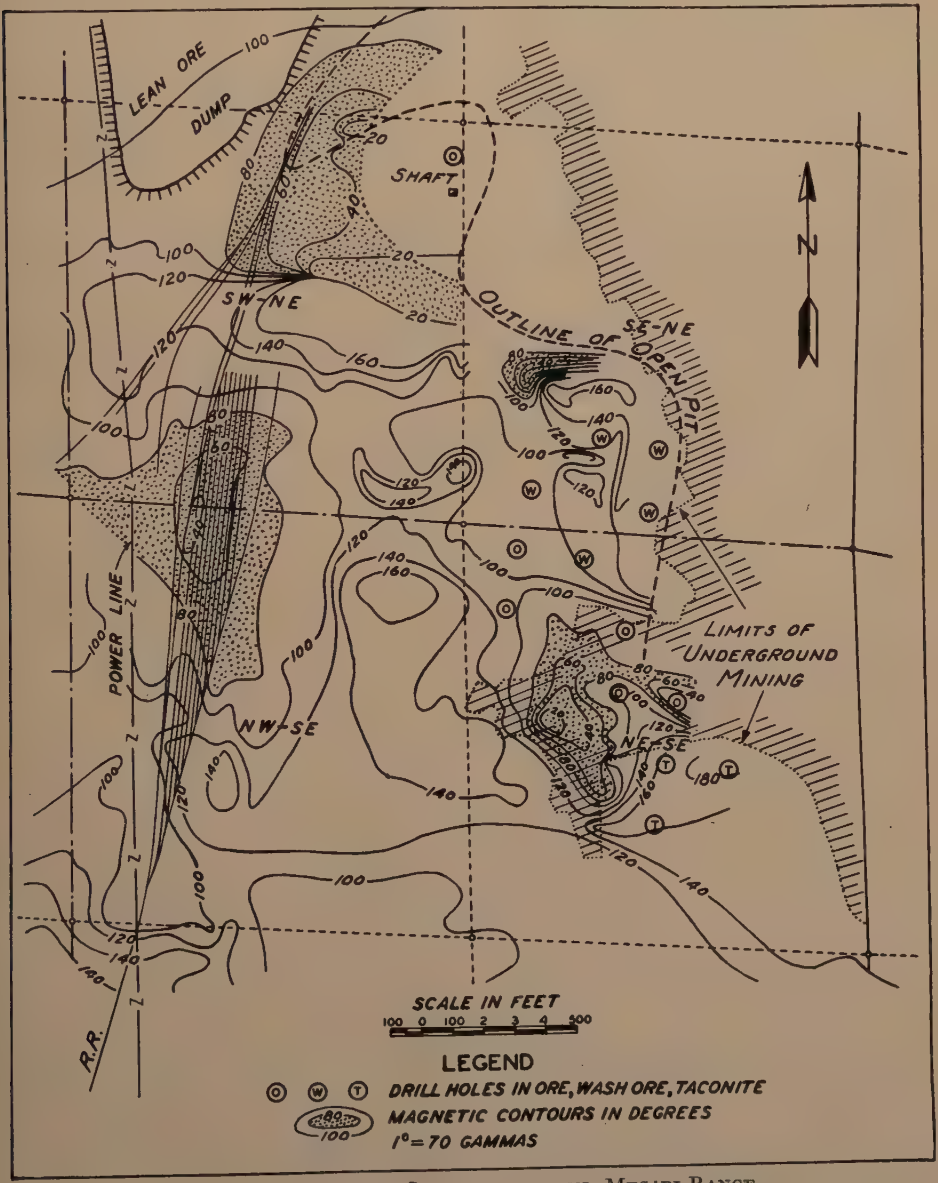

FIG. 4.—RESULTS OF SUPERDIP SURVEYS, MESABI RANGE.

Superdip survey, the results of which are contoured on Fig. 4.

The ore adjacent to the old shaft, beneath about 400 ft. of taconite, is outlined by a magnetic low surrounding the shaft area.

Another magnetic low area, southwest of the shaft and away from the wash ore, is in a location which, from surface conditions, would be favorable for a shaft. This second "low" also outlines 15 closely spaced railway tracks, a scrap-iron pile, a high-tension transmission line and other man-made impediments to a magnetic survey. This second "low" area was explored by a vertical drill hole which did not encounter ore. The new shaft is located near by.

This is an example where surface culture did not bother the resistivity survey except possibly for some stray currents from the power line and mine plant. Moisture-laden horizons underground also may have influenced this work. To balance the score, moisture and stray electric currents did not affect the magnetic survey. One inference is clear, the ideal place to run a geophysical survey is in virgin territory not contaminated by man.

To complete this example: The magnetic survey gave valuable data on the wash-ore horizon and was used as a basis to select the most desirable shaft location. Drill holes were put down to confirm conclusions based on the geophysical surveys and the new shaft is now being put up as a raise from underground.

Another problem, not so complicated, was a Superdip survey around an open pit to outline the possible limits of the ore body and also to aid in selecting a shaft location safely away from the ore area. The Superdip did a good job in showing high magnetic areas where no ore occurred. After selection of the most favorable shaft location from an operating standpoint, an exploration hole was drilled to confirm the magnetic work before the shaft was sunk.

At another property the open pit was getting too deep for railway grades, which tie up ore in banks. Some of the bottom ore of this deposit and also that of a near-by ore body must be removed by underground methods. A shaft with surface layout and stock-pile ground was needed and the desired location was between the operating open pit and a drilled ore reserve about ¼ mile away. A Superdip survey showed a magnetic-high zone between the known ore bodies, confirming some scattered drilling, which indicated the absence of ore. The new shaft and mine layout are nearly completed.

### *Example 4.—Tracing a Fissure Ore Body in an Area Underlain by Additional Ore at Depth*

Fig. 5 shows an area underlain by the lower cherty and lower slaty horizons, with the upper cherty horizon in the southern portion of the map.

An ore zone beneath the paint-rock layer (intermediate slate) underlies much of this area. This ore in the upper part of the lower cherty horizon is developed and it is being mined underground.

A channel or fissure ore body from 60 to 150 ft. deep, and trending south 15° east had been mined on the northwest 40 acres shown on the map. To the north this ore occurred immediately below glacial drift. Three drill holes proved that the ore extended into the territory to the south. One of the holes encountered 65 ft. of glacial drift and an additional 45 ft. of taconite before reaching ore.

The purpose of the Superdip survey was to determine if possible the continuation and trend of this ore body to the south. The underlying flat ore body at a depth of about 200 ft. complicated the magnetic picture. Power lines in the vicinity had only a local effect.

On Fig. 5 the 65° magnetic contour is shown by heavy dashed lines as it coincided with the limits of the ore body as mined to the north and shown by full lines. The mined-out part of the fissure ore body is crosshatched.

The coincidence of the ore boundaries indicated by magnetic contours and determined by underground mining now in progress is remarkable.

## CONCLUSIONS

The results of Superdip surveys demonstrate that it is sometimes possible to

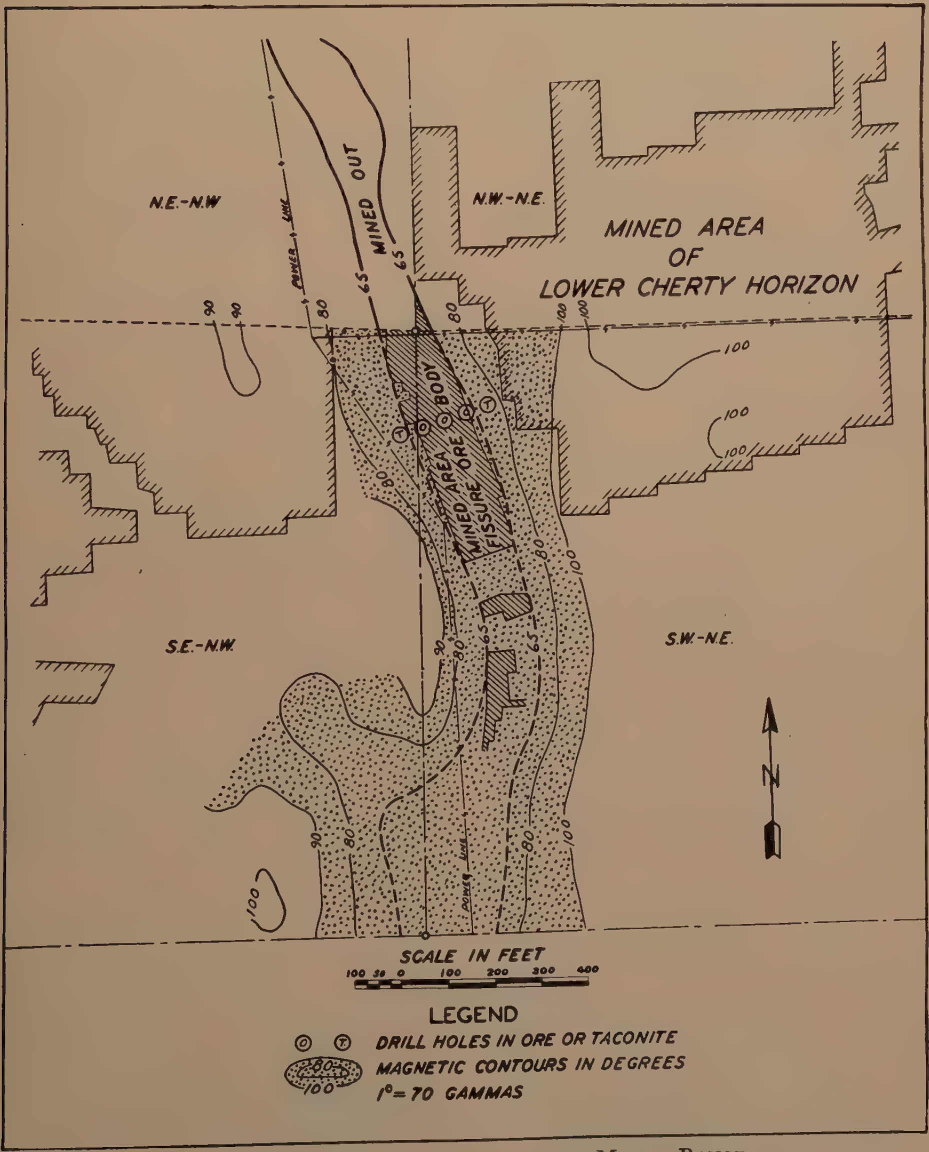

FIG. 5.—RESULTS OF SUPERDIP SURVEYS, MESABI RANGE.

distinguish areas underlain by ore from those underlain by iron formation on the Mesabi Range by means of magnetic surveys. This is because ore bodies are well oxidized and yield lower magnetic readings than are obtained from unoxidized iron formation from which the ore originated.

In areas where the ore has formed from magnetite-bearing horizons, the magnetite is either partially or entirely oxidized into hematite and limonite and as a result, the magnetic attraction of such ore is lower than the parent taconite.

It follows that in a given area, the greater the magnetic content in the iron formation, the greater the difference there will be between the magnetic readings obtained over ore and those obtained over the iron formation associated with it.

Conversely, in areas where the taconite contains but a small amount of magnetite, the difference in magnetic readings will not be as clearcut and it may be difficult or impossible to distinguish those areas underlain by ore from those underlain by taconite, by such a method.

Furthermore, the overlying Virginia slate is characterized by low magnetic readings, and where patches of this slate or even Cretaceous slate overlie taconite, the results of a magnetic survey can be misleading. In addition, the Duluth "gabbro" at the eastern end of the range contains closely associated anorthosite and gabbro phases and these have extreme variations in magnetic properties.

The Superdip is efficient in outlining areas where no ore occurs, but which might be amenable to magnetic concentration.

In summary, on the Mesabi Range the magnetic results obtained are dependent upon the preoxidation magnetite content of the particular horizon of iron formation being tested; upon the subsequent degree of oxidation therein; upon the nature of the overlying and underlying horizons, and upon other factors as illustrated by examples presented in this paper.

The Hotchkiss Superdip has demonstrated its value as an aid in seeking iron ore and guiding drill exploration. For best results, all available geological data should be considered when interpreting the magnetic anomalies obtained.

## ACKNOWLEDGMENT

All of the Superdip surveys described in this paper were run by members of the Geological Department of the Oliver Iron Mining Co. Acknowledgment is due the officers of the Oliver Iron Mining Co. for permission to use the information presented.

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## DISCUSSION

J. W. GRÜNER.*—The paper by Dr. R. H. B. Jones is a very timely one. As the very large high-grade ore bodies become exhausted it becomes increasingly more difficult to outline accurately the smaller ones or the marginal positions of the large ones without some new methods. The Superdip, as Jones shows so well, is about of the proper sensitivity for the Mesabi range.

Another very pertinent fact brought out is the great irregularity found in the oxidation of the magnetite beds. The magnetic layers are of chief interest to companies that contemplate the recovery of magnetite from the unoxidized taconite. As I have also emphasized recently (The Mineralogy and Geology of the Taconites and Iron Ores of the Mesabi Range, Minnesota, Iron Range Resources and Rehabilitation, St. Paul, Minn., 1946†), this oxidation is so erratic and widespread that the greatest caution will have to be exercised in outlining magnetic ore bodies even in the Lower Cherty division, where the chances are very much better than in the Upper Cherty. It is questionable whether the Superdip would be a distinct help there unless we learn a great deal more about the interpretation of magnetic profiles of flat-lying formations. Fortunately the contacts between magnetic and oxidized taconite are commonly sharp, as illustrated so well by Fig. 5 of Jones' paper. This secondary oxidation is absent, of course, on the East Mesabi Range, where magnetic layers are very persistent.

ROBERT B. SOSMAN*—When artificial precipitated magnetite is oxidized at a temperature in the vicinity of 100°C, the oxide is converted into a chocolate brown ferric oxide of the formula $Fe_2O_3$, free from ferrous iron, yet possessing the same magnetic susceptibility as the original $Fe_3O_4$. This form of $Fe_2O_3$ is variously called "ferromagnetic ferric oxide," "maghemite," or "gamma ferric oxide," and has been found as a natural mineral. If a body of magnetite in the Biwabik were to oxidize to this form, the oxidized and leached ore body might successfully conceal itself from Mr. Jones' survey by this ferromagnetic pretense.

So far, gamma ferric oxide has not been found on the Minnesota iron ranges. I thought I had found some in the Hibbing pit in 1940, in a strongly magnetic specimen of soft ore differing in color from the prevailing hematite red, but a laboratory test showed that it contained finely divided residual magnetite and not gamma ferric oxide. The best test is to heat the specimen to 650 to 750°C, whereupon the gamma form will quickly revert to ordinary hematite, which is paramagnetic and uninfluenced by a steel hand magnet.

It is rather surprising that the gamma form has not been found more frequently where magnetite has been oxidized to martite and hematite. Its occurrences seem confined to places where a solution containing ferrous and ferric iron has precipitated and oxidized, as in the gossan of a copper ore body.[16] It is possible that "magnetite" is not always the same mineral, and that some kinds are oxidizable to gamma ferric oxide and others are not.

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* Department of Geology, University of Minnesota, Minneapolis, Minnesota.

† Obtainable without charge from the Commissioner of Iron Range Resources and Rehabilitation, State Office Building, St. Paul, Minnesota.

* Formerly Research Laboratory, U. S. Steel Corp., Kearny, N. J.; now Professor of Ceramics, Rutgers University, New Brunswick, N. J.

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# Radioactivity Exploration with Geiger Counters

BY HENRY FAUL*

(San Francisco Meeting, February 1948)

## ABSTRACT

MEASUREMENT of radioactivity of rocks and ores has developed into a complete method of geophysical exploration. The problem falls into three natural categories: (1) surface radiation measurement in the field and underground; (2) radioactivity logging of drillholes; and (3) laboratory analysis of samples.

Portable counters are used in the field for tracing radioactive formations and for quantitative estimates of radioactive content of surface rock. Wartime advances in battery design make it possible to construct small precision counters that can be carried easily. Even the smallest conventional diamond drillholes can be logged, and actual content of the radioactive element can be measured. The exact grade of very thin active layers or veins can be estimated only approximately. Samples may be analyzed for their alpha, beta or gamma activity, and beta assaying is the best suited to rapid routine operation. Simple calibration with standard samples is sufficient for rough work, but a density correction is necessary when greater precision is required. Samples with high emanating power should be fixed and stored for three weeks to reestablish the radium-radon equilibrium prior to precision assaying. Carnotite can be fixed by sintering or molding with plastics. Among the important new developments, a low-voltage Geiger tube has been designed recently, and crystal scintillation counters hold great promise for high-efficiency gamma-ray measurement.

## INTRODUCTION

During the last few years we have witnessed an unprecedented rise in the demand for radioactive minerals, coming as a direct consequence of the discovery of nuclear fission and subsequent developments, the full impact of which is by no means in sight even now. It remains certain, however, that radioactive minerals, and especially those which can be used as ores of uranium and thorium, will continue to be of great interest not only to the prospector and mine operator but also to the various inspection parties envisioned by most of the schemes advanced for the control of nuclear energy on a world-wide basis.

It is clear, therefore, that the measurement of radioactivity in rocks in the field as well as in samples in the laboratory has become a method of geophysical exploration of some importance. If we consider the recent technological advances in the general fields of nuclear physics and electronics it seems proper to review the present status of the methods and procedures of radiological exploration.

## GENERAL CONSIDERATIONS

### *Equilibrium*

If we take a fixed amount of uranium that has just been purified, some of it will immediately start decaying into its first daughter element which in turn decays into its own daughter and so on down the line until the last daughter is reached—a stable isotope of lead. The rate at which a radioactive element decays is frequently expressed as the *half-life*, which is the length of time required for the decay of half the atoms in a given amount. The half-life is inversely proportional to the *decay constant* $\lambda$ which is defined as the number of atoms which decay in one unit of time divided by

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Manuscript received at the office of the Institute May 5, 1948. Issued as TP 2460 in MINING TECHNOLOGY, November 1948.

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the total number of atoms in the sample of the decaying element.

As the decay cascade proceeds, the isotopes with a small $\lambda$ will tend to accumulate, and those with a very large $\lambda$ will decay a very short time after their birth. We can visualize the process if we imagine water running through a series of vats connected in cascade by pipes of different sizes. The water accumulates and builds up pressure in the vats with small outlet pipes but runs right through the vats with large drains.

We can easily see that if we start with all but the top vat empty (the case of purified uranium), the water level in each vat will tend to approach a relative equilibrium position. When this mark is reached, each vat is slowly emptying and flow throughout all is at the same rate. The bottom vat, which has no outlet,* is slowly filling up. Obviously then, the rate at which water is running from each vat into the next lower one must be proportional only to the amount of water in the top vat at any one time as long as equilibrium exists.

The concept of equilibrium becomes very important when we try to measure the amount of any particular radioactive element in a rock by measuring the rate of overall radiation. We must remember that radioactive parent and daughter are always chemically different and can therefore be separated in nature by normal geological processes. It is as if one or more of our vats had sprung a leak and we could no longer be sure that the flow from any particular vat to the next lower one was the same as the flow from the top vat.

Such a case arises when we measure the radiation from some of the supergene uranium minerals (carnotite, tyuyamunite, and others). One of the daughter elements of uranium is radon, a noble gas, and these minerals are sufficiently porous to permit the escape of some of the gas with the consequent loss of radiation from radon and its numerous daughter elements (Fig 1). Therefore, if we compare the radioactivity of an impermeable mineral such as pitchblende with the radioactivity of carnotite, the carnotite will appear low even though the uranium content of the two minerals be the same. The ability of a mineral to lose radon is called the *emanating power*[1] and may vary widely from specimen to specimen. This effect may be a serious source of error in radiological sample testing, unless the samples are treated to fix the gas in place and then stored for a few weeks to reestablish the radon equilibrium.

Another important ratio is that between $U^{238}$ and radium which is almost always constant in nature. There are a few instances, however, where minerals are undergoing intense leaching or chemical weathering that results in differential movement of the active elements. In this way the equilibrium may be disturbed faster* than it can reestablish itself and anomalous assay results are obtained from radioactivity measurements. The question of the radium-uranium ratio in carnotites has been investigated by Lind[1,3] and coworkers who found the ratio constant in large samples even though individual specimens showed various degrees of deviation from the average value.

### *Naturally Radioactive Elements*

Most of the naturally occurring radioactive elements belong to three major families each of which comprises isotopes of most of the elements on the heavy end of the periodic table. The parent elements of the three families are uranium (${}_{92}U^{238}$ and ${}_{92}U^{235}$) and thorium (${}_{90}Th^{232}$), and the daughter elements include isotopes of protoactinium, actinium, radium, francium, radon (agar), polonium, bismuth,

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* The final product (lead) is stable.

[1] References are at the end of the paper.

* 320,000 years are required to establish 95 pct equilibrium between uranium and radium[2] (see Fig 2).

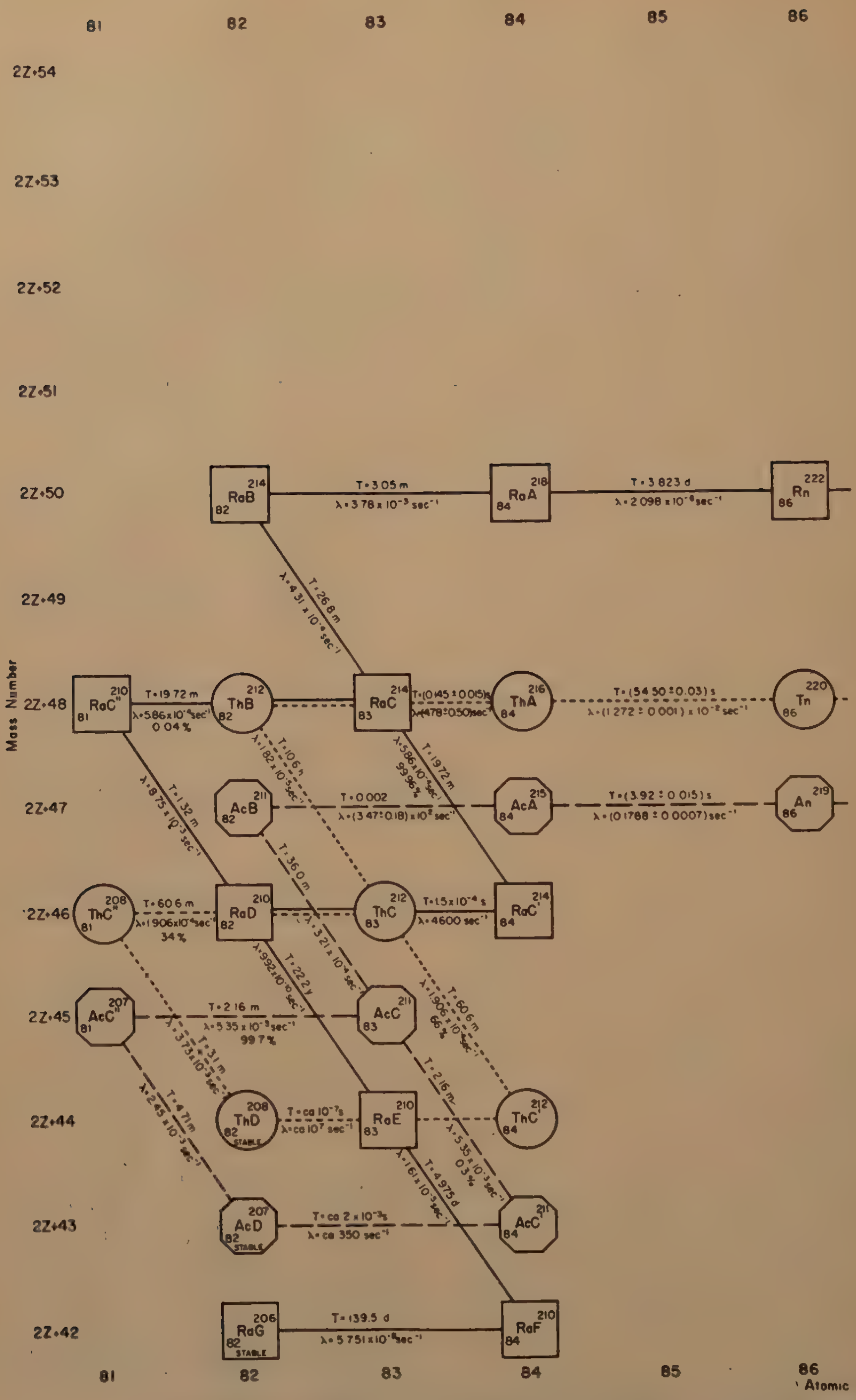
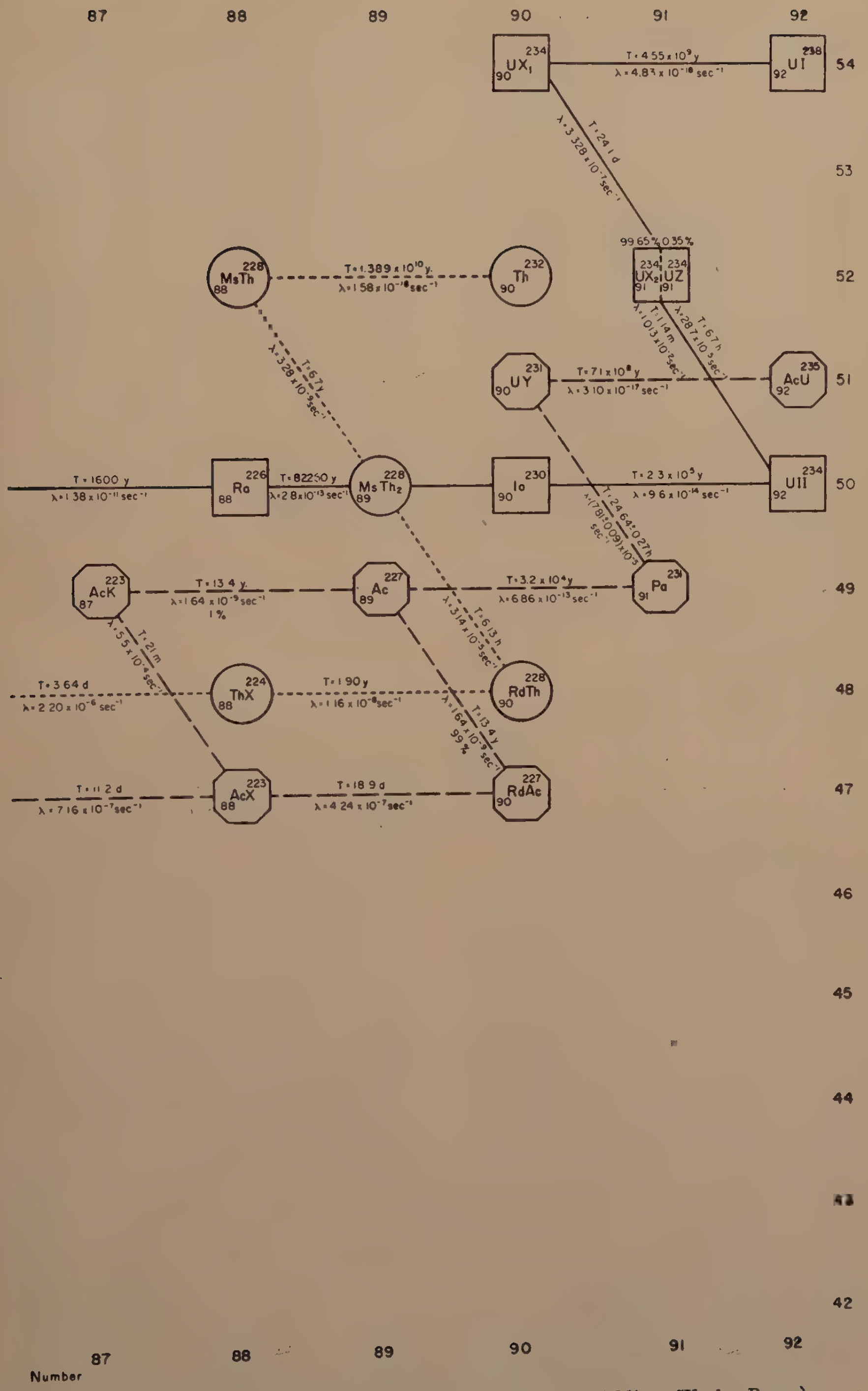


FIG 1—DECAY SCHEMES AND CONSTANTS OF URANIUM AND THORIUM SERIES.



(*From* The Science and Engineering of Nuclear Power,[7] *courtesy Addison-Wesley Press.*)

lead, and thallium (in order of decreasing atomic weight). The decay cascade of the three families together with half-lives and decay constants is shown graphically in Fig 1.

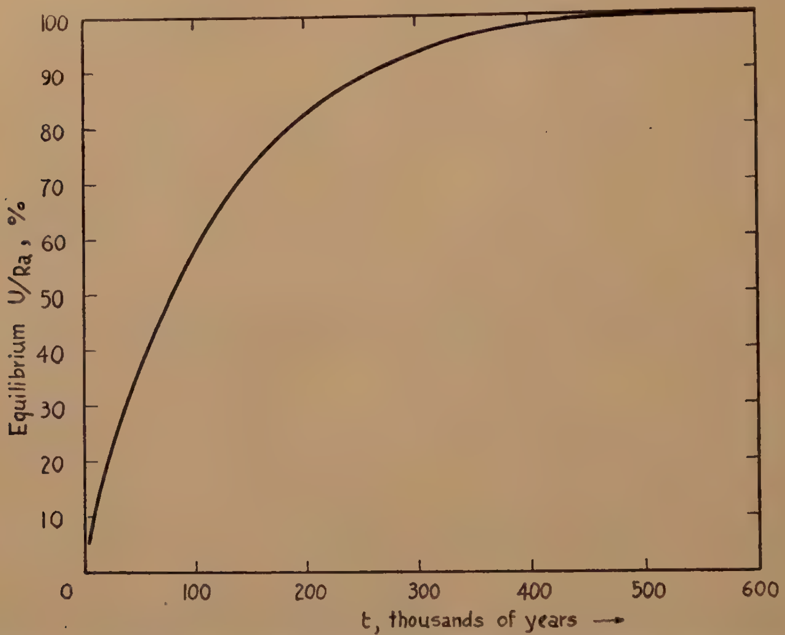


Fig 2—Uranium-radium equilibrium plotted as function of time. (*After Kovarik.*[2])

The two uranium families always occur together in nature with the parent elements intimately mixed in a fixed proportion which is constant throughout the earth ($U^{235}/U^{238} = 1/139.2$). The thorium family is sometimes found with uranium, but more frequently occurs in minerals by itself. Most of the commercial ores of uranium contain no thorium whatever.

There are other naturally radioactive elements: the alkalis potassium and rubidium, the rare earth samarium and lutecium, and the heavy metal rhenium.

The radioactive isotope of potassium ($_{19}K^{40}$) comprises 0.012 pct of all natural potassium and has a half-life of $1.4 \cdot 10^9$ years.[4] It emits beta rays, orbital electron capture X rays and gamma rays,[5] and occasionally may be the principal contributor to the radioactivity of a rock. Natural potassium compounds can be quantitatively analyzed by geophysical means with great accuracy.[6] The activity of common $K_2O$ amounts to roughly 27 betas per second per gram. The activity of the uranium series (in equilibrium) is approximately 40,000* betas per second per gram of $U_3O_8$. Consequently, a very small uranium contamination may cause a large error in a potassium analysis, but the error caused by potassium in uranium and thorium analyses is usually negligible.

Rubidium receives its radioactivity from $_{37}Rb^{87}$ which has an isotopic abundance of 27.2 pct in natural rubidium and a half-life of $5.8 \cdot 10^{10}$ years or about 20 times the accepted age of the earth.[7] Common $Rb_2O$ has a specific activity of roughly 10 times greater than the activity of common $K_2O$.

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* Neglecting the weak betas of $UX_1$ and RaD.

The rubidium beta, however, is ten times weaker than that of potassium.* As a chemical element, rubidium is comparatively rare in nature. Its richest "ore" is lepidolite which usually contains about 1.5 pct $Rb_2O$, and lesser amounts are found in amazonite, pollucite, and some varieties of microcline and muscovite.

The radioactive samarium isotope is $_{62}Sm^{148}$ which makes up 14.2 pct of natural samarium. It is the only known alpha emitter outside of the heavy elements, but the half-life of the activity is extremely long.† The element itself is found only in minute quantities in samarskite, cerite, monazite, and a few other rare minerals. The lutecium activity also has a long life‡ and the concentration of the radioactive isotope ($Lu^{176}$) is small.§ The element itself is very rare and usually associated with yttrium in nature. The recently reported beta activity of rhenium[11] probably comes from $_{75}Re^{187}$ which has an isotopic abundance of 61.8 pct. The reported half-life is immense, and the radiation very weak.‖ Small amounts of rhenium are found in columbite, tantalite and wolframite.

It is clear that rubidium, samarium, lutecium, and rhenium, although interesting in themselves, are not commonly encountered in radioactivity exploration, and are therefore of little importance within the scope of this paper.

### *Radiation and Absorption*

Each of the radioactive isotopes emits an alpha or a beta particle as it decays. Gamma rays and X rays are also emitted, the latter arising from orbital electron capture. We can think of the alphas as stripped helium atoms and the betas as accelerated electrons. An alpha particle weighs almost 8000 times as much as a beta, and consequently travels more slowly and has an intense ionizing effect along its path. It is, therefore, very easily absorbed even though its kinetic energy may be very great. A sheet of paper or a thin foil of metal will stop all natural alphas. The smaller (and faster) beta particle requires a fairly thick sheet of metal to be stopped completely, and all but the weakest beta rays can pass through a thin window of mica, glass, or aluminum. The gamma ray is an electromagnetic radiation of very short wavelength, equivalent to very "hard" X rays, and has a considerable penetrating power. There is a general tendency, however, to overestimate the range of gammas in rocks, and we shall see later that out of a given large number of energetic ("hard") gamma rays only very few will get through a layer of rock one foot thick. Gamma-ray measurements therefore have a comparatively short range compared with some other geophysical methods.

### *Statistical Error*

An ionizing event in a counter, namely, the passage of a beta particle or the production of a photoelectron by the passage of a gamma ray, is not influenced by events that preceded it nor by events that may follow. The counts, therefore, occur at random (or "perfectly irregular") intervals.

In any random procession, the number of events that occur in one unit of time is not the same as the number of events in the following unit of time, and the fractional difference between a single reading and an average of a large number of readings is referred to as the "statistical error." This quantity will be different for each individual measurement, and it cannot be given a definite numerical value. Instead, we must think of it in terms of probability.

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* Maximum beta energies: Rb — 0.13 Mev, K – 1.3 Mev (Segré Chart[8]). Rb also emits weak gamma rays.

† $1.4 \cdot 10^{11}$ years (Segré chart.[8])

‡ Libby[9] reports $7 \cdot 10^{10}$ years for the half-life of lutecium.

§ 2.5 pct in natural lutecium.[10]

‖ Approximately $3 \cdot 10^{12}$ years, 0.04 Mev.[11]

The absolute probable error* of a measurement is defined as

$$p = 0.6745\sqrt{N}$$

where $N$ is the total number of counts recorded. We have an even chance that a measurement will differ from the average of a large number of measurements by an amount smaller than $p$. In other words, if we take a large number of repeated measurements with a perfect instrument, half of the readings will differ from the average by an amount smaller than $p$ and half will differ by more than $p$. It is important to realize that $p$ is a function of $N$ only, and it does not depend on the time it took to make the measurement. The probable error can therefore be kept constant in a series of radioactivity measurements, by measuring the time required to rack up a given number of counts rather than the number of counts per unit of time. This procedure is commonly used in routine testing of radioactive samples (See Fig 10).

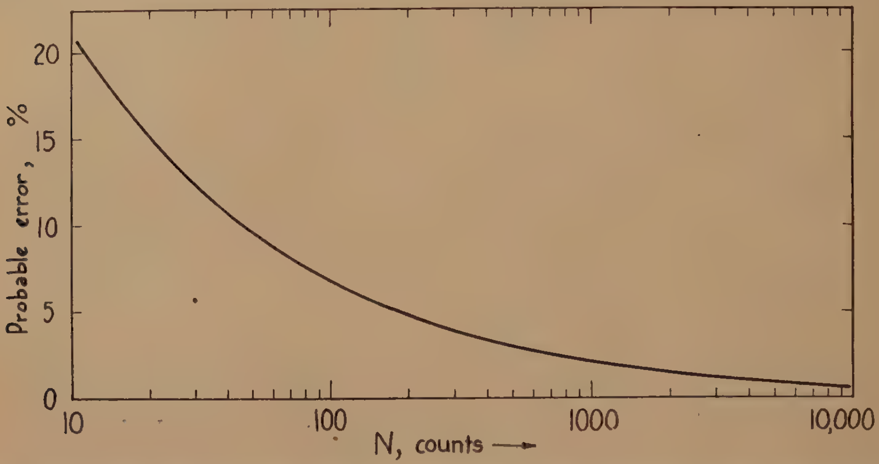

FIG 3—PROBABLE ERROR (IN PER CENT) AS FUNCTION OF TOTAL NUMBER OF RECORDED COUNTS.

## METHODS OF EXPLORATION

The instruments and procedures of radioactivity exploration fall into three distinct groups: (1) surface intensity measurement; (2) borehole logging; and (3) laboratory analysis of samples.

The three methods represent three different approaches to the same problem: the geological evaluation of relative or absolute radioactivity distribution data. In this sense, the methods overlap and one may be used as a check against the other. We shall take up, one by one, the procedures and instruments of each of the three methods.

### *Surface Intensity Measurement*

### *(Traversing)*

It is common procedure to measure surface radiation intensity of a given area at a number of points which are surveyed and plotted on a map. Radioactivity contours then may be drawn, or the data can be plotted in "sections," showing radiation intensity versus horizontal distance (Fig 4). Considerable geological information usually can be obtained from plots showing only relative radiation intensity, expressed in some arbitrary units (e.g., counts per minute), but it is often desirable to calibrate such a map to show the actual concentration of the radioactive material in the surface layer of rock. The most frequently used method of calibration involves channel sampling at selected points and laboratory analysis of these samples. The relative-intensity map is then used to interpolate between the sampled points.

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* The commonly used percentage probable error is the ratio of the absolute probable error and the measured quantity, expressed in per cent (see Fig 3).

Although it is obvious that quantitative results are best obtained in areas of bare rock, traversing may be useful in regions covered with overburden or areas where radioactive bodies may exist at shallow depths. Russell[12] showed that even in some "areas of deeply weathered residual soils the stratigraphy of the underlying formations may be mapped by plotting the areal variations in the radioactivity of the weathered products," and Ridland[13] clearly recorded the No. 3 ore-zone at LaBine Point, N.W.T. where a vein of pitchblende comes within 60 ft of the surface by detecting the radiation from radon gas carried along the otherwise barren fissure above the vein (Fig 4). On the other hand, very misleading measurements may be made in regions covered by thick glacial drift and in the uranium mining districts, where much high-grade ore is frequently scattered about the countryside by blasting and spillage from ore trucks.

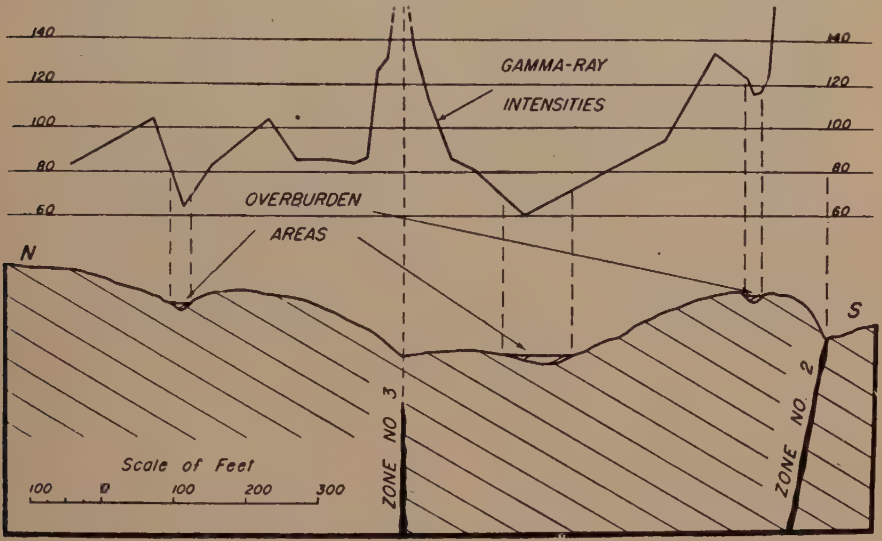

Fig 4—Surface radioactivity profile through ore zones at LaBine Point, N.W.T. (*After Ridland*[13]).

*Portable Instruments*

Portable counters have, of course, been built for a long time, but the main stumbling block in their design has always been the difficulty in securing a stable supply of high voltage for the Geiger-Müller tube. The lowest voltage conveniently used with conventional modern G-M tubes is about 800 v. Various types of oscillators have been used to supply this voltage, but their output must necessarily be kept small to conserve battery power, and proper voltage control therefore becomes difficult. Consequently, oscillator-powered high-voltage circuits are not well suited to accurate quantitative work in the field, although they may be quite sufficient for rough qualitative detection.

The obvious source of stable high-voltage is a dry-cell battery but, until the war, dry cells were not available in sufficiently small sizes to be practical for this purpose. Rajewsky[14] in Germany made a battery that weighed only about 1 g per volt, and a similar dry-cell was designed and mass-produced for the proximity fuze program in this country. In its present commercial form, a 900-v battery weighs 3 lb 3 oz and occupies 59 cu in.

The simplest counter circuit then may consist of the G-M tube, the battery, and a pair of sensitive earphones. Instruments of this type are useful for rough work, and because of their simplicity are very rugged, stable, and completely reliable.

Auditory counting, however, is very tedious and limited to low counting rates. The human ear cannot reliably follow more than about 150 to 200 random counts per minute, and the human error is great even at very low counting rates. Therefore, it is desirable to eliminate the necessity of counting by ear either by amplifying and recording the Geiger pulses mechanically with an impulse register (relay-driven clock) or by metering the average current flow resulting from the passage of amplified pulses. Impulse recorders are most useful for low rates (up to about 300 counts per minute) and rate meters are usually used for higher activities.

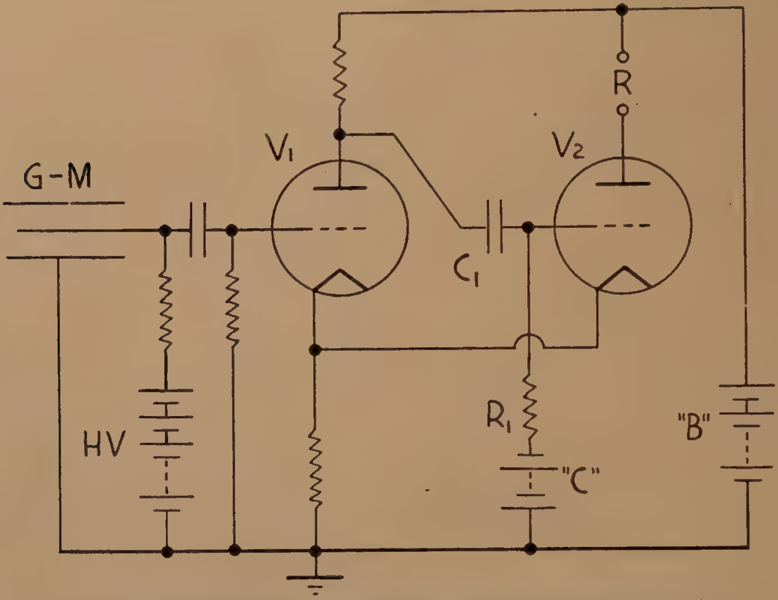

Fig 5—Circuit for portable counter. (*Modified after Schmitt.*)
Symbols: G-M—Geiger-Müller tube, $V_1$ first stage (normally on), $V_2$ second stage (normally off), *R* recorder, *HV* high voltage battery, "*B*," "*C*" plate and bias batteries.

A cathode-coupled multivibrator circuit* (Fig 5) makes a very good "amplifier" for Geiger pulses because a small trigger pulse fed into the circuit produces a much more powerful "shaped" output pulse. The effective gain per tube is considerably higher than that obtained from a conventional amplifier, resulting in economical power drain which in turn permits the use of small batteries.

The multivibrator may be adapted to drive a commercial impulse register by using an R-F pentode (such as the 1N5) and a power amplifier tube (1T5, 1Q5, or similar) for $V_1$ and $V_2$, respectively (Fig 5). The register is connected at *R*, and a 22½ v "C" battery provides bias to cut off the power tube. Most counting rate meters, on the other hand, use subminiature tubes for $V_1$ and $V_2$, for these tubes require as little as 15 milliwatts of filament power per tube. A microammeter is then connected at *R*, by-passed by a high-capacity elec-

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* Also known as Schmitt's trigger circuit.[15] For an extensive discussion of trigger circuits see O. S. Puckle.[16]

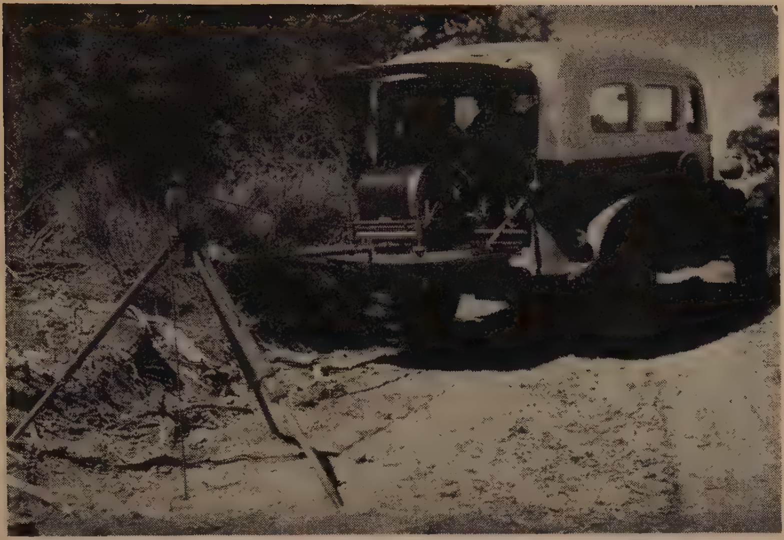

FIG 6—TRUCK MOUNTED COUNTER LOGGING A 1½-IN. DIAMOND DRILL HOLE.
The drum holds 200 ft of cable in one layer, but holes up to 800 ft deep can be logged. The truck carries storage batteries, a charger, and 200 ft of casing which is set and pulled by hand.

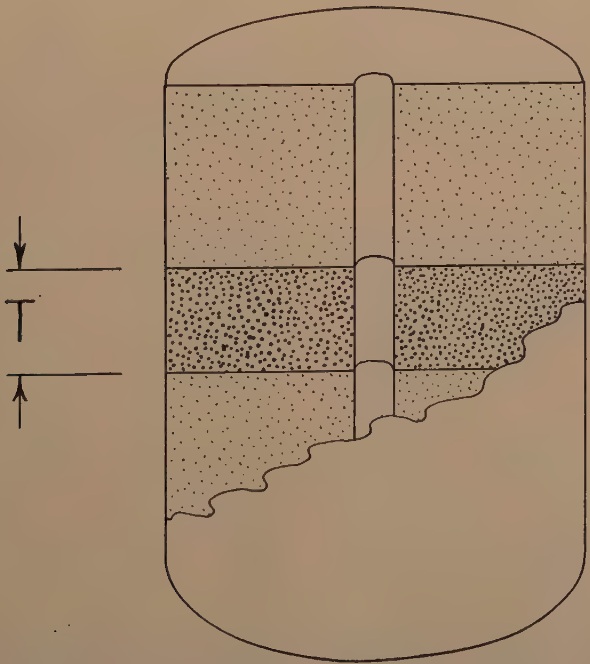

FIG 7—"SYNTHETIC DRILL HOLE" FOR CALIBRATION OF LOGGING INSTRUMENTS.

trolytic condenser to form a storage circuit which smooths out the output pulses to give a steady current proportional to the number of pulses per unit of time. The "C" battery is omitted when subminiature tubes are used, but provision is made to vary the coupling constants $R_1$ and $C_1$ in two or three steps so that a single instrument may have several ranges of sensitivity.

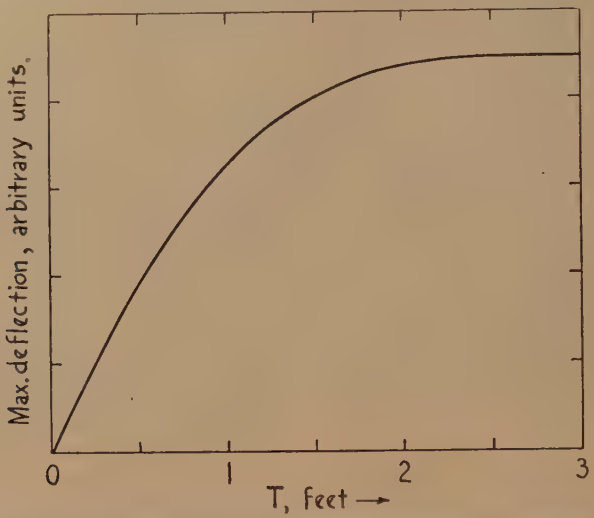

Fig 8—Maximum meter deflection plotted as function of thickness of a uniform horizontal bed of radioactive rock, drilled vertically and logged with an 8-in. Geiger counter.

### *Borehole Logging*

Well-logging by radioactivity is now an established process in the petroleum industry[17] but, until recently, the large (3 to 5-in. diam) well-logging instruments could not be readily adapted for use in the small diamond drill holes commonly drilled for mineral exploration. In its simplest form the logging instrument (Fig 6) usually consists of a counter and pre-amplifier housed in a probe and lowered into the hole. A chart-recording counting rate meter is generally used at the surface to give a continuous log of radiation intensity versus depth. The smallest conventional diamond drill holes are 1.48 in. in diameter, and quite shallow, compared to oil wells. The holes may frequently be drilled in bad ground and it may be necessary to case them before the probe can safely be lowered, so that the probe itself should be smaller than 1 in. overall outside diameter. Modern subminiature tubes, small high-efficiency pulse transformers, and hearing-aid type construction make it possible to make a probe about ⅞ in. in diameter

Relative radioactivity data are all that is required for well-logging where the logs are used primarily for stratigraphic correlation, but in mineral exploration work absolute calibration is of the utmost importance. It is essential to know as much as possible about the actual concentration of radioactive matter in the immediate vicinity of the hole. The theoretical approach is not simple but it is possible to calibrate a probe empirically by surrounding it with radioactive material of known concentrations arranged in layers of different thickness to form a "synthetic drillhole"[18] (Fig 7). The material must be chosen so as to have a gamma-ray absorption coefficient similar to the actual rocks expected in the field.

We may imagine a bare uniform outcrop of radioactive rock with a gamma-ray intensity $I_o$ at the surface. Let us now cover this outcrop with a uniform layer of rock of thickness $x$. The gamma radiation intensity will be reduced to a new value $I$, which will be given approximately by the relation:

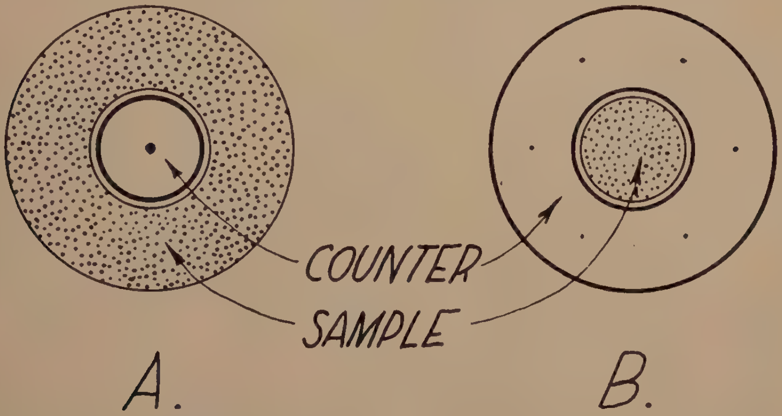

FIG 9—RELATIVE POSITIONS OF SAMPLE AND COUNTER FOR GAMMA RAY MEASUREMENT.

$$I = I_0 e^{-\mu \rho x}$$

where $\mu$ is the *mass absorption coefficient*, $\rho$ is the density and $e$ is 2.72 (the base of natural logarithms). The mass absorption coefficient decreases with increasing energy of the gamma rays, but it is roughly independent of the composition of the absorbing material. For one-Mev gamma rays $\mu$ is approximately 0.06 and we may take 2.5 as the density of the absorbing rock. If the thickness $x$ is 30 cm, we then have

$$\frac{I}{I_o} = e^{-4.5} = 1.1 \text{ pct}$$

In other words, a foot of rock will absorb about 99 pct of the gamma rays passing through it.*

It turns out that with an 8-in. G-M tube in sandstone the total indication of the counting rate meter is proportional to the absolute concentration of radioactive matter around the tube only when the radioactive layer (see Fig 7) is over 2 ft thick and laterally continuous for more than a foot or two. In thin layers of "ore" the factors of *thickness* and *grade* cannot be separated by analysis of readings taken with a single G-M tube, for a 2-in. layer of 2 pct ore will produce essentially the same effect as a 4-in. bed of 1 pct ore. In either case, the peak meter deflection will be substantially smaller than it would be for a 2-ft layer of the same grade. The relationship of deflection versus thickness for a given grade is shown graphically in Fig 8. The curve levels off at about 2 ft and continues flat for thicknesses greater than 2 ft. This distance will be somewhat less for shorter G-M tubes, and somewhat greater for longer ones, but it is usually impractical to make small tubes shorter than about 4 in. or longer than a foot or so. Nevertheless, approximate information on the thickness of thin radioactive beds can be obtained by logging a drill hole first with a long G-M tube and then relogging the radioactive zones with a very short tube. The relative widths of the deflection peaks will serve as an indication of the thickness of the beds.

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* It follows from the same relation that roughly 2 in. of rock will reduce one-Mev gamma ray intensity by a half.

### *Laboratory Analysis of Samples*

The radioactive content of rocks and minerals has been studied by many workers using a variety of methods: Evans and Goodman[19] measured the radium content of rocks by a direct fusion method of releasing the radon and determining the total ionization effect of the gas and its immediate decay products in an ionization chamber. A method for direct alpha-activity measurement of samples was developed by Finney and Evans,[20] Keevil and Grasham,[21] Evans and Goodman,[22] and Nogami[23] using layers of radioactive mate-

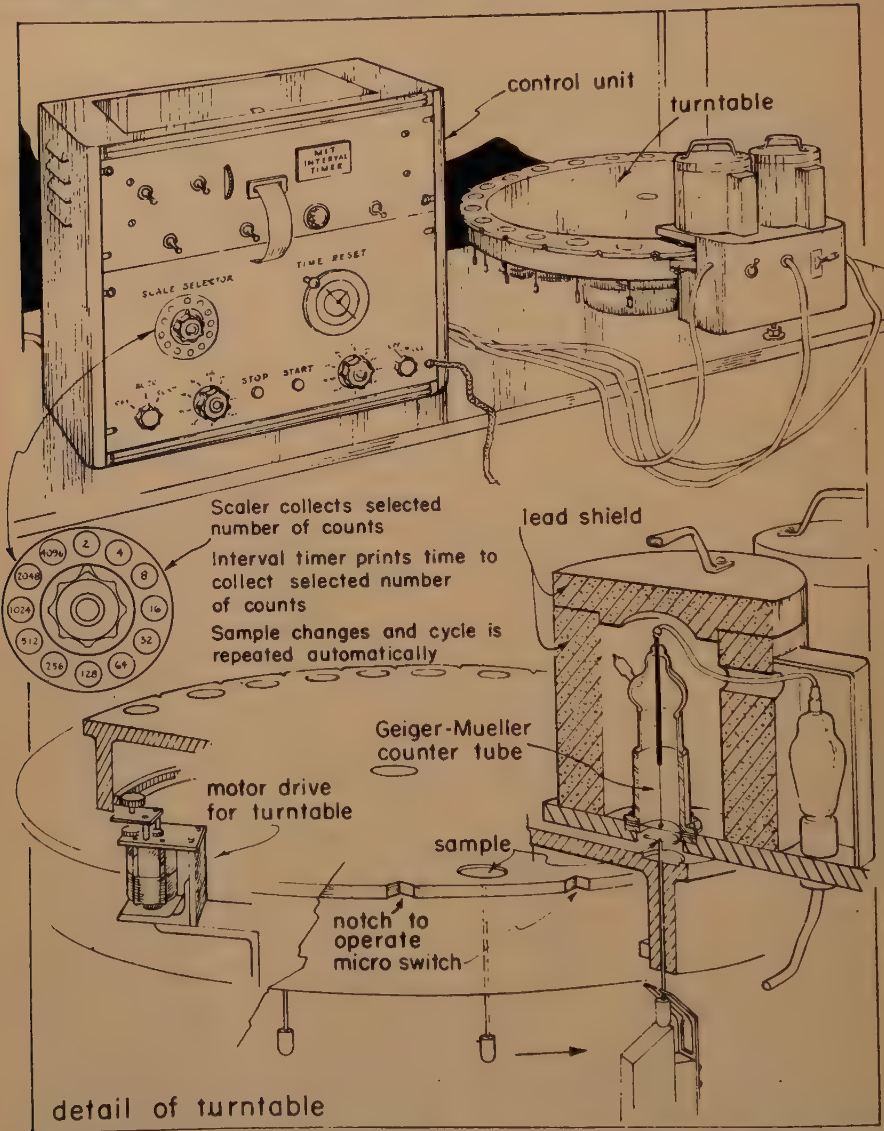
control unit
turntable
MIT INTERVAL TIMER
TIME RESET
SCALE SELECTOR
STOP
START
2
4
8
16
32
64
128
256
512
1024
2048
4096
Scaler collects selected number of counts
Interval timer prints time to collect selected number of counts
Sample changes and cycle is repeated automatically
lead shield
Geiger-Mueller counter tube
motor drive for turntable
sample
notch to operate micro switch
detail of turntable

FIG 10—FULLY AUTOMATIC SAMPLE TESTER RECORDING THE TIME REQUIRED TO COLLECT A SELECTED NUMBER OF COUNTS.

rial considerably thicker than the range of alpha particles in the material.

The fogging of photographic plates by radioactive minerals, unwittingly discovered by Becquerel,[24] was more recently suggested as an analytical method for radioactive elements by Tyler and Marais[25] who studied the radioactivity of minute mineral grains by embedding them in a photographic emulsion. The autoradiography of rocks and minerals in polished sections was brought to a high degree of perfection by Yagoda.[26]

The Geiger counter offers a convenient method for measurement of beta and gamma radiation of samples. Pontecorvo[27] placed his samples coaxially around a thin-walled counter which unshielded was primarily sensitive to betas, but could quickly be changed into a gamma counter by interposing a sheet of metal between sample and counter (Fig 9*a*). The geometry of such an arrangement is very favorable,* but a fairly large amount of sample is required. A scheme suggested by Geiger and Müller[29] (for a different purpose) can be used when only small samples (a few grams) are available. It consists essentially of a multi-anode counter "wrapped" coaxially around the sample (Fig 9*b*). For rapid beta-ray analysis of fairly active materials, such as low-grade ores of uranium or thorium, the bell-type mica-window counter[30] is very successful. Mica windows can be made very thin* so that the counters are sensitive even to very soft betas. The gamma efficiency, on the other hand, can be made low by using a stepped center wire, so that the counting region is limited to the vicinity of the window and the secondary electrons produced by gammas in the upper parts of the counter do not reach the counting region. Automatic sample changers[31] are used efficiently with the bell-type counters. A fully automatic setup used at M.I.T. is shown in Fig 10.

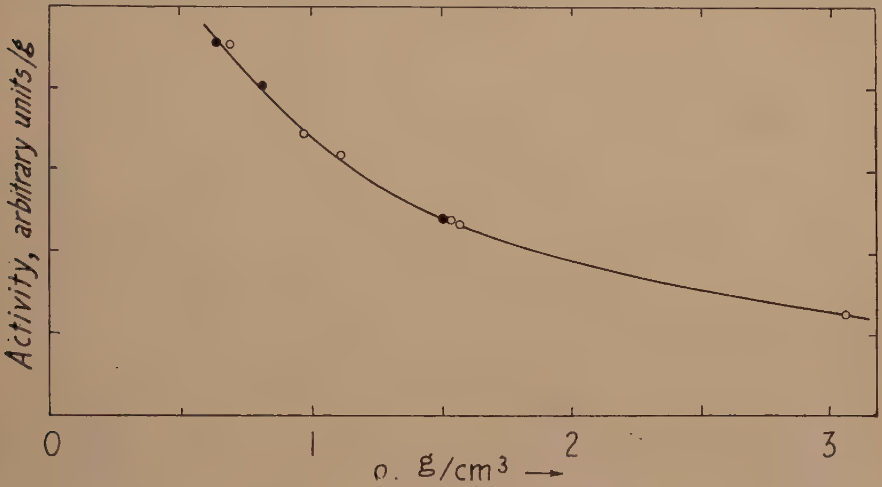

FIG 11—BETA COUNTER CALIBRATION CURVE, SHOWING DROP IN SPECIFIC ACTIVITY WITH INCREASING DENSITY OF SAMPLE. (*Solid circles after Beers.*[33])

### *Calibration*

Because of the uncertainties in the energy distribution of beta-ray spectra and beta-ray absorption coefficients, theoretical calibration of beta counters can be only very rough. An empirical calibration is therefore essential.[32] Beta counting of samples of known radioactive content and a various types of rocks, and the simple standard-sample calibration will give reasonably accurate results when judiciously applied.

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* The same geometry was used by Beers and Goodman,[28] and for the analysis of potassium solutions by Barnes and Salley.[6]

* Counters with windows that weigh as little as 1.5 mg per cm² are now commercially available.

For more accurate work, however, it is usually desirable to make a density correction. The effect of variable apparent density can be established very simply by diluting, in a given proportion, a known amount of radioactive material with radioactively inert minerals of various densities. We can then plot specific activity against apparent density of the various mixtures, and we find that the points fall on a smooth curve (Fig 11). Knowing the apparent densities of our unknown samples, we can now adjust their measured beta activities to any arbitrarily established "mean density" by using the curve. The shape of the curve

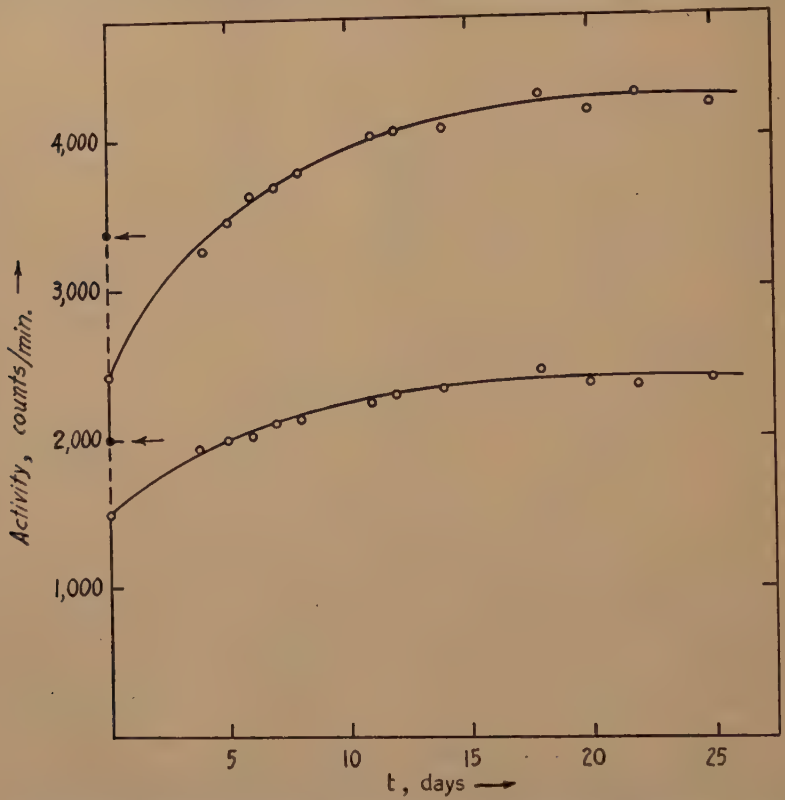

FIG 12—INCREASE IN THE BETA ACTIVITY OF CARNOTITE SAMPLES AFTER SINTERING. THE ORIGINAL ACTIVITY (BEFORE HEATING) IS INDICATED BY ARROWS.

statistical comparison of the beta activities of these "standard" samples to the activities of unknown samples does not, in general, constitute sufficient calibration, unless all samples are essentially identical in their apparent density,* and in the proportion of radioactive elements contributing to the total activity, which also implies constancy of the factors of radioactive equilibrium and emanating power. This condition may be met, to a greater or lesser extent, in is virtually independent of the absolute grade of the samples. The points measured by Beers[33] (solid circles, normalized at density $\rho = 1$) are in good agreement with those measured by the author, even though the average grade of Beers' samples* was several hundred times lower than the grade of the writer's samples.

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* Apparent density is here defined as the weight of the actual sample divided by its volume when placed under the counter.

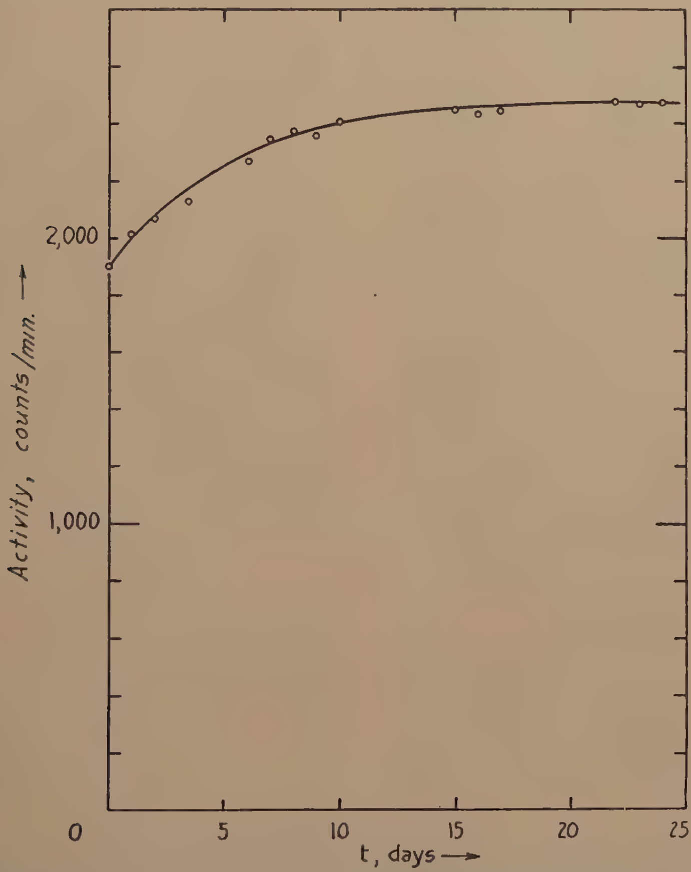

FIG 13—INCREASE IN BETA ACTIVITY OF CARNOTITE SAMPLE AFTER MOULDING INTO BUTTON WITH VINYLITE.

Samples containing minerals with high emanating power require additional treatment to bring back the radioactive equilibrium disturbed by loss of radon. It is necessary to render the sample impermeable in some way to stop the outward diffusion of the gas and then store the fixed sample for about two to four weeks to reestablish equilibrium. Radon has a half-period of 3.82 days (see Fig 1) so that 90 pct equilibrium is reached in 12 days and about 25 days' storage is required for 99 pct equilibrium. Probably the simplest way of "sealing" a sample is by sintering, i.e., heating to a temperature where the emanating mineral begins to fuse. Fig 12 shows the beta activity of two samples of low-grade carnotite that have been sintered at about 1000°C for one hour, plotted against time in days. Although heating drives off most* of the radon, the sintering lowers the emanating power to a very small value, and equilibrium builds up in complete agreement with the theoretical rate. It is interesting to note that apart from a slight discoloration, sintering produces no change visible to the naked eye in most powdered carnotite samples. Only rarely is it necessary to regrind the sample after sintering. The overall loss of water which results from the sintering process must be considered in computing the grade for some samples, although the correction is usually very small.

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* The uranium content of Beers' samples was on the order of 1 to 10 × $10^{-6}$ g per g, but the author's samples contained about 0.6 pct uranium.

Another technique for arresting the radon loss consists of molding the powdered sample with a plastic resin such as lucite or vinylite. The activity of the molded buttons is then measured along with standard samples prepared in the same way. The molding treatment is obviously more laborious than sintering, but it is necessary for samples that do not respond satisfactorily to sintering. Fig 13 shows the results of molding a carnotite sample with high emanating power.

The calibration of gamma counters is complicated by the fact that the geometrical relation of the sample to the counter must be taken into account. The question has been analyzed by Davis[35] and by Hushley and Dixon,[36] who present a comprehensive theoretical treatment of the problem. The final calibration curve depends on the constants of each particular instrument, but it will be essentially similar to the curve shown for beta calibration.

## Future Developments

It has frequently been said that the Geiger counter is obsolete as a radiation detector. This statement is not entirely unjustified, but at the present time there is little doubt that Geiger-Müller counters are by far the simplest and most reliable portable instruments for field use. A counter capable of operating at a voltage of only 150 v has been developed by Simpson[37] but it is not yet available commercially. By eliminating the need for high voltage, the new counter will certainly affect the future design of portable instruments by further reducing their size and weight.

The mica-window beta-counter has no peer at the time of writing, and none appears in the offing. However, important developments in gamma-counting technique have been announced recently. Experiments of Kallman[38] and many other investigators[39–43] have shown that various organic and inorganic crystals* emit light in the ultraviolet region when traversed by ionizing radiation. The crystals are transparent to their own light so that the flashes can be "seen" and counted by an electron-multiplier phototube. The effect is particularly useful for the detection of gamma rays, and the overall efficiency can be made ten to fifty times better than that of the best G-M counters. The electronic equipment necessary for the scintillation counter is somewhat more complex than the comparatively simple G-M pulse amplifiers, and there are various complications of technique, but it seems certain that this new device will replace the G-M counter in many applications where high gamma counting efficiency is required.

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* R. D. Evans,[34] found that 1600°C was required to completely de-emanate a sample of carnotite ore.

* Naphthalene, anthracene, phenanthrene, scheelite, fluorite, alkali halides, and other widely different materials have been used successfully. Various slight impurities increase the intensity of the light flashes.

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# A Review of Fluorescence as Applied to Minerals, with Special Reference to Scheelite

By John W. Vanderwilt,* Member A.I.M.E.

(Chicago Meeting, February 1946)

## Abstract

The fluorescence of scheelite has been an important aid in recent years in the discovery and development of scheelite deposits. The use of fluorescence of synthetic compounds in industry, particularly in advertising, also has expanded greatly during the past decade. As a result, a number of new and convenient units for producing ultraviolet energy have become available, which have led to a greater application of fluorescence in the study of minerals.

Studies have shown that the fluorescent color of a mineral may vary: (1) with the kind and quantity of trace elements, and (2) with the particular wave length of ultraviolet energy used. The wave length is dependent first on the type of generating equipment used, and finally on the filter employed to remove the visible light that may be present. A number of so-called ultraviolet lamps are listed and described, together with suitable filters known to be available.

Scheelite fluoresces brilliantly with wave lengths of 2200 to about 3000Å., but not with longer wave lengths. The most convenient and efficient sources of ultraviolet for scheelite fluorescence are portable and laboratory units designed particularly for use with scheelite. So-called ultraviolet light globes of the photoflood type emit wave lengths too long to cause scheelite to fluoresce.

Attention is called to the minerals that fluoresce in a manner similar to scheelite, which necessitates a check test by means of characteristic physical properties. Chemical tests are the final authority both for identity and content of tungsten in scheelite ore.

## Introduction

Recent development and extensive use of fluorescent lighting make it unnecessary to explain what fluorescence looks like. The fluorescent light tube is in reality a mercury vapor arc, which generates ultraviolet energy. Coating the inside walls of the tube is a substance, called a phosphor, that converts the ultraviolet rays to visible rays of light. In a like manner, certain minerals will transform ultraviolet energy to visible rays of various colors. The phenomenon of brilliant fluorescent colors cannot be satisfactorily described or reproduced photographically; a display must be seen to be appreciated.

Only a few years ago fluorescence was regarded primarily as a source of entertainment, chiefly in spectacular displays of specimens in museums. The brilliant fluorescence of willemite, a zinc ore mineral, at Franklin, New Jersey, was utilized quite early in checking the efficiency of certain milling operations. Fluorescence has been used extensively in exploring for scheelite, as well as in mining this ore, during the past 10 years.

The use of fluorescence in detecting scheelite, an important tungsten ore mineral, is well established, and much success has attended this new method; scheelite is a very inconspicuous white to gray mineral in ordinary light, but it

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Manuscript received at the office of the Institute July 25, 1945. Issued as TP 1967 in Mining Technology, March 1946.

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fluoresces conspicuously. However, as is generally true with any method, it is not foolproof. Different minerals may show similar fluorescence and all varieties of the same mineral do not fluoresce alike, and the various wave lengths of ultraviolet energy may produce very contrasting effects on a particular mineral. Therefore some precautions are necessary when fluorescence is used in the identification of minerals. Scheelite fluoresces brilliantly with the shorter ultraviolet energy emitted by a cold quartz mercury-vapor tube, but it is unaffected by the longer wave lengths obtained from the Purple X globe and other ultraviolet sources utilizing similar filter glass.

### How Fluorescence Is Produced

Fluorescence is a type of luminescence, or the emission of visible light by a substance when exposed to ultraviolet rays. Other types of luminescence are thermoluminescence (temperature), triboluminescence (friction), X-ray luminescence, chemiluminescence, bioluminescence, and so forth. Ultraviolet energy is by far the most effective excitant, and it causes fluorescence not only of various minerals but also of artificial inorganic and organic compounds, certain mosses and lichens, and even some varieties of lizards, worms, and insects.

Fluorescence is the result of a conversion of invisible ultraviolet energy to visible energy or light. It is generally believed that the molecular structure of a substance plays an important part in the conversion, but the mechanics whereby invisible energy is transformed to light has not been fully explained.

Most substances fluoresce only as long as they are exposed to the required stimulating energy, but a few substances continue to fluoresce from several seconds to hours, depending on the material, after the stimulating energy is shut off. This phenomenon is known as phosphorescence.

Artificial fluorescent materials are called phosphors. These usually consist of an inert substance with a small quantity of a substance or element called the activator, which is necessary to produce fluorescence. Common mineral phosphors are silicates, tungstates, and sulphides of calcium, barium, or strontium, with copper, lead, bismuth, zinc, or manganese as the activator. Organic compounds that fluoresce are relatively common, of which quinine sulphate and petroleum are well-known examples.

Optimum brightness of fluorescence in most minerals requires only a relatively small concentration of ultraviolet energy, but a particular wave length or narrow band of wave lengths is essential. The most effective wave length must be determined for each mineral.

### Uses of Fluorescence

Fluorescent lighting has become so important that it deserves special mention. For the first time it is possible to produce a cool light that closely approximates daylight, and this high quality of lighting costs only about one third as much in electric energy as conventional lighting. The versatility of fluorescent lighting comes from the fact that each particular phosphor produces a characteristic fluorescent color or fractional part of the visible spectrum. Thus, in order to duplicate daylight, or any other desired color, it is only necessary to select and mix in proper proportions a number of phosphors with the required color values.

Fluorescent dyes and pigments are being used in advertising and theatrical displays. Manufacturers use fluorescent dyes in products to permit future identification. Inks that fluoresce are used to prevent and detect forgeries. Fluorescence is used in criminology, medicine, biology, and pharmacy. Certain quantitative analyses are possible by use of fluorescence; uranium in particular is amenable to this method. The fluorescence of natural oil is utilized for detecting potential oil-producing horizons in oil-well drilling. These and many other practical applications of fluorescence have been developed during the past 10 to 15 years, and further advances, as well as new uses, are to be expected in such a new field.

## Reason for Conflicting Reports as to Fluorescent Colors of Minerals

The use of fluorescence of minerals has advanced with the development of various convenient sources of ultraviolet energy, which are now available at reasonable cost. However, ultraviolet units differ in the wave lengths they produce. In addition, there are variations peculiar to particular minerals. These two factors are not always fully appreciated or understood, and therefore some of the resulting contradictory claims as to fluorescence is not surprising.

The light condition under which fluorescence is observed, and color standards also, must be taken into account. Total darkness is required if fluorescing colors are faint or if close comparisons are to be made, as with the scheelite-powellite analyzer mentioned later. With more casual observations, surprising results can be obtained in a partly darkened room, although in all instances total darkness is recommended.

## Nature of Ultraviolet Energy

Light rays are energy in the form of wave motion. This energy varies as to wave length and frequency of vibration, but it has a constant velocity of about 186,300 miles per second. Each wave length has its own definite properties, chief of which is color as seen by the eye. The eye, however, is limited to what it can see; red being the longest and violet the shortest wave lengths that are visible. Wave lengths slightly longer than red are termed the infrared or heat rays, and those shorter than violet are the ultraviolet rays.

The common unit of measure of light rays is the millimicron ($10^{-6}$ mm.), but the Ångstrom unit (Å), one ten-millionth of a millimeter ($10^{-7}$ mm.), is used to define ultraviolet wave lengths. The visible spectrum varies from about 8000Å. maximum for red, to 4000Å. minimum for violet, and the ultraviolet rays are a continuous series from about 4000 to 100Å., or less, merging at the short end of the series with X-rays.

In the visible spectrum a difference of 200 to 400Å. is represented by each of the several colors red, orange, yellow, green, blue, and violet; and the average eye is capable of detecting still smaller differences through recognition of intermediate hues. In the ultraviolet region the various wave lengths also show marked contrasts; for example, 2650Å. is an effective germ-killing center, 2967Å. is the peak for erythemal effectiveness, and 3650Å. is a center for photographic and fluorescent effects. Fluorescence, however, is not limited to the 3650Å., and differences in the wave length of ultraviolet energy may or may not produce contrasting fluorescent colors in various minerals; scheelite does not respond to near ultraviolet energy, but it fluoresces conspicuously on exposure to shorter wavelengths.

Thus, before the significance of an observed fluorescent color of a mineral can be fully appraised, it is essential to know at least approximately the wave length of the ultraviolet energy that produced the fluorescence.

The ultraviolet region is subdivided by authors of technical papers into the near ultraviolet (4000 to 3000Å.), middle or intermediate ultraviolet (3000 to 2000Å.), and the far or extreme ultraviolet (2000 to 100Å. or less). However, the dividing limits are not always used consistently. The terms "long" and "short" ultraviolet are in common use, particularly in semitechnical papers, but the defined limits vary from 3000 to 3500Å. therefore these terms are not satisfactory except for denoting very general or relative values.

The nature of the ultraviolet radiations available to the average person, who does not have access to the special equipment necessary to isolate specific wave lengths, consists of two differing though overlapping regions of the ultraviolet spectrum. One range of wave lengths is from 3100 to 4000Å., but with 3650Å. predominating; and the other range is from 2300 to 4000Å., with 2537Å. predominating.

The longer range of wave lengths, 3100 to 4000Å., is obtained from incandescent filament globes and various mercury units that operate at moderate to high pressures and at relatively high temperatures. The common glass ordinarily used screens out the wave lengths under 3200Å., and even where special glass is used the heat-resisting filters necessary to remove visible light also removes wave lengths of 3100Å. and less. The second type of ultraviolet radiations is obtained from the cold quartz mercury tube, which permits the use of a non-heat-resisting filter designed for transmitting wave lengths in the broad range of about 2300 to 4000Å.

## Sources of Ultraviolet Energy

Sunlight contains ultraviolet rays in varying proportions, dependent on the clearness of the atmosphere that tends to filter them out. Under favorable conditions, and in direct sunlight, the practical minimum wave length is about 3100Å. However, sunlight is not a convenient source of ultraviolet energy, because best light is limited to a few hours at noon, and it is dependent on variations in the atmosphere and weather.

Incandescent-filament light globes generate ultraviolet energy somewhat in proportion to their operating temperature. Thus photoflood globes yield more than ordinary globes, but the glass used for the globes screens out radiations shorter than 3200Å.

In recent years photoflood types of globes made with red-purple filter glass have been available on the market under trade names such as Wonderlite, Black Bulb, and Purple X. The Purple X globe at the rated 115 volts has a transmission range of from 3100 to 4300Å., with 3650Å. predominating. This globe, which fits a standard light socket, is rated 250 watts, 115 volts, and a 50-hr. life with intermittent burning. Globes of this type are a most convenient source for the small quantities of near ultraviolet energy necessary for fluorescent effects on minerals.

The 2½-watt argon glow lamp is another convenient source for a limited output of ultraviolet energy; and, having a relatively low output of visible light, it can be used without a filter if fluorescence is relatively strong. However, filters improve the fluorescent colors. The predominating wave length is around 3300Å.

A high output of ultraviolet radiations required for lighting foyers, theater auditoriums, murals, etc., is obtained with various types of mercury light units requiring special transformers and filters. Mercury lamps originally designed solely for ordinary lighting are made with low to high globe pressures and temperatures, depending on the results desired. Some types of high-intensity mercury lamps have an inner globe of quartz glass and an outer envelope of heat-resisting red-purple filter glass. Such units with filters are designed for production of near ultraviolet energy, usually above 3200Å., with their maximum output around 3650Å.

High-tension spark discharges using iron, nickel, tungsten, or other metal electrodes generate a fairly continuous spectrum of ultraviolet energy, with wave lengths as low as 1000Å., depending on voltages as well as the kind of metal used in the electrodes. This source is quite popular in laboratories, as the shorter wave lengths can be isolated with suitable filters.

The electric arc is an excellent source of both short and long ultraviolet energy, and it gives a continuous spectrum, which is an advantage in certain types of work. A disadvantage of an arc arises from the high operating temperatures, as these require heat-resistant filters, which do not have high efficiency for short ultraviolet transmission.

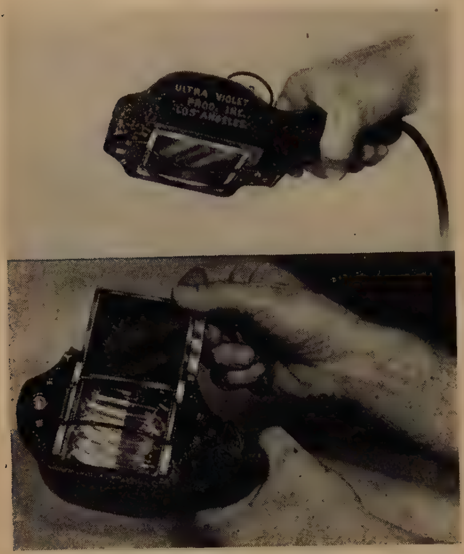

FIG. 1.—HOUSING WITH FILTER FOR QUARTZ TUBE (ABOVE) AND WITH FILTER REMOVED (BELOW) TO SHOW CLOSELY FOLDED QUARTZ TUBING.

The cold quartz tube utilizes ionization of mercury vapor at low pressures and temperatures to produce ultraviolet energy, 60 to 90 per cent of which is in the 2537Å. range, with smaller quantities around 2652, 3126, and 4047Å., and some in the visible spectrum.

A cold quartz tube contains a small globule of mercury and an atmosphere of an inert gas, commonly argon, to facilitate lighting. A current applied to electrodes sealed in at each end of the tube produces the arc through the mercury vapor. This tube has two outstanding advantages in addition to the relatively large quantity of ultraviolet energy at 2537Å. that it produces. First, the tube consumes very little electric energy, so that with a suitable transformer it is practical to operate small units from standard dry-cell batteries. Second, in the absence of even moderate heat, a non-heat-resisting filter like Corning's Red Purple Corex A-9863 can be used, which transmits the shorter wave lengths with high efficiency.

True quartz tubes are quartz glass. Some manufacturers add a flux to the quartz to lower the fusion point and decrease cost of production, but the fluxes limit the passage of shorter wave lengths, and such tubes are less useful than those made of pure cast or fused quartz.

Germicidal tubes operate on much the same principle as the cold quartz tube, with a transformer designed to operate on standard 110-volt current. The composition of the glass tube, a product of the Corning Glass Works, has not been made public; but the glass transmits ultraviolet wave lengths as short as 2537Å., and it is a competitor for fused quartz. With suitable filters (Corning No. 9863) germicidal tubes produce fluorescent colors similar to those produced by the cold quartz tube, and if the cost of large filter glasses is not prohibitive the germicidal tube should be suitable for illuminating relatively large areas necessary for displays of sizable mineral collections.

An important characteristic of the mercury vapor arc which should be kept in mind is the nature of the spectrum produced. The spectrum is not a continuous one, but it consists of a series of ultraviolet resonance bands, the most important of which have wave lengths around 2537, 2652–2655, 2967, 3126–3132, 3341, 3650–3662, and 4047Å.; and, as previously stated, 60 to 90 per cent of the energy, depending on the exact specifications of the quartz tube, will be in the 2537Å. band. So-called black light from incandescent lamps and high-temperature mercury vapor tubes, because heat-resisting filters must be used, are limited to the ultraviolet resonance bands 3126–3132, 3341, 3650–3652, and 4047Å.

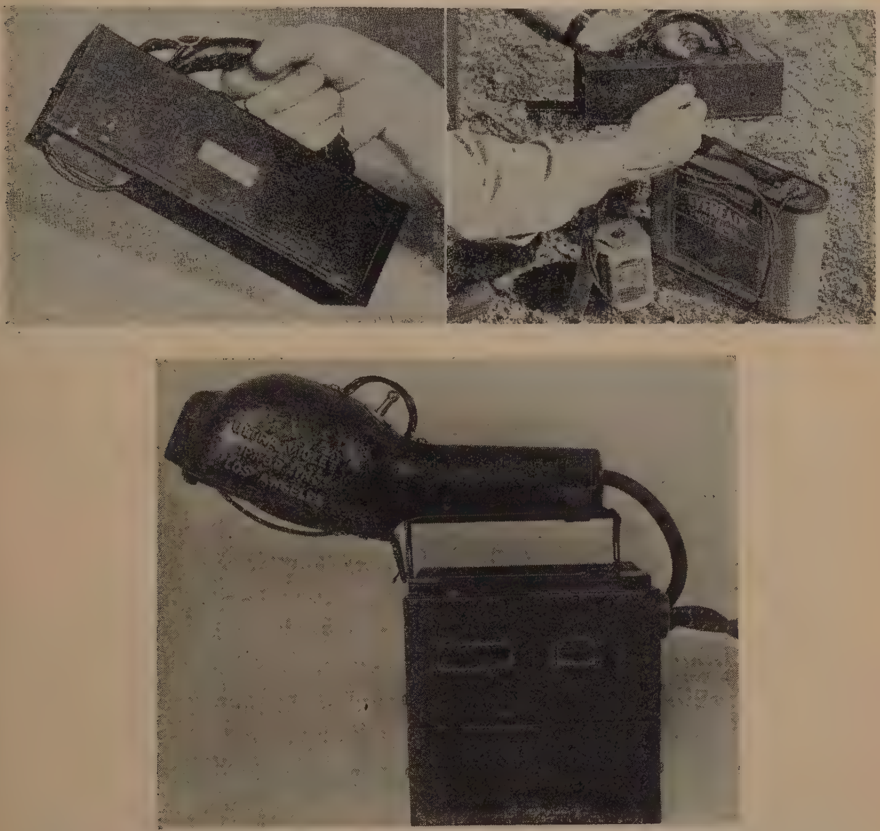

FIG. 2.—TWO POPULAR MODELS OF MINERALIGHT, PORTABLE ULTRAVIOLET UNITS.
The small model (above) has a compartment for use of a standard electric lantern battery, and it also has a plug-in socket for operation on a hot-shot battery. The larger model is for a six-volt hot-shot battery, and the lamp housing (Fig. 1) is detachable from the lower unit.

Very little is known of the fluorescent effect of wave length intermediate to the mercury vapor spectral bands, and no work has been done, as far as the writer knows, on the composite effect of various combinations of narrow bands of ultraviolet energy.

A number of portable and semiportable lamps using cold quartz tubes are available on the market at prices ranging from $35 to $85. Reflectors are, or should be, polished aluminum, which is the most efficient for reflecting ultraviolet wave lengths. The ultraviolet output is dependent on the length of the quartz tubing, which is closely folded so as to concentrate as great a length as possible in a given area, as shown in Fig. 1. The life of a cold quartz tube of this type is said to be 2000 to 4000 hr. of continuous operation. Fig. 2 shows two popular models, which the writer has found very satisfactory. The smaller unit measures 9¾ by 2¾ by 2¾ in., and weighs only 3¾ lb. with battery, and is good for 6 to 8 hr. operation. The larger unit has about 50 per cent greater ultraviolet illuminating area and operates 30 to 40 hr. on a 6-volt hot-shot battery. The units have self-contained transformers for operation with the batteries, and special transformers are available for operation on standard 110-volt currents. The best fluorescent effects are obtained at 12 to 18 in., covering areas about 2 to 3 ft. across, but strongly fluorescent minerals glow conspicuously in total darkness at distances of several feet.

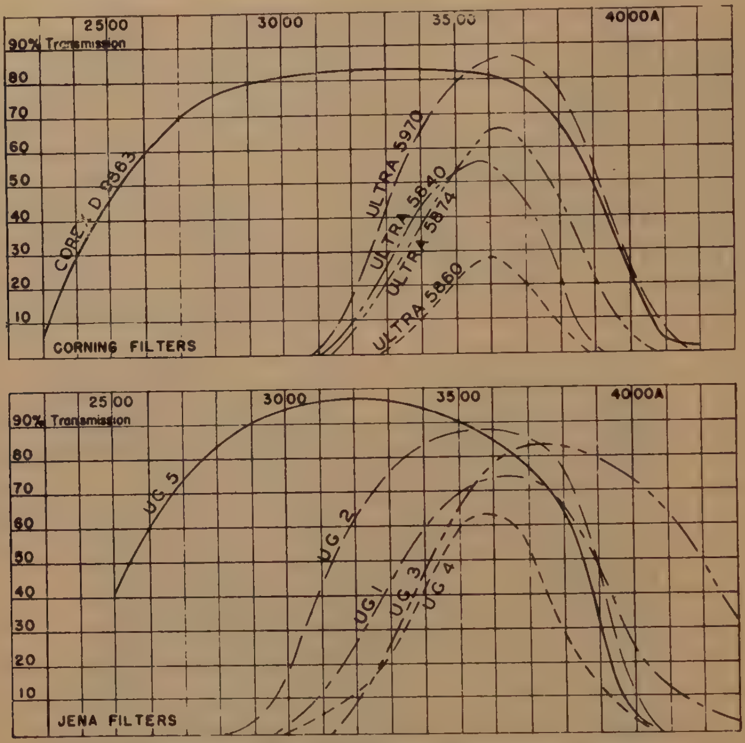

Fig. 3.—Transmission curves of various ultraviolet glass filters.
*a.* Data by Corning Glass Works, Corning, N. Y.
*b.* Data by Fish-Shurman Corporation, New York, N. Y.

## Filters

A filter is required to remove the visible light produced with ultraviolet, because light tends to mask fluorescence. Common glass, regardless of its color, is opaque to wave lengths under about 3200Å. Quartz glass will not only pass ultraviolet but also visible light. High-temperature ultraviolet sources, such as the electric arc and hot mercury vapor tubes, add to the complication, because they require heat-resisting filters. Various materials, including liquids, gases, and plastics, are used as filters for special purposes, but they are less practical for general use with minerals. The Corning Glass Works markets a series of ultraviolet filters easy to obtain and satisfactory for most purposes. The Jena Glass Works was the outstanding European manufacturer of ultraviolet filters. Fig. 3 shows the transmission curves of the Corning and Jena filters for comparison of their similarities and differences, as well as their mutual limitations.

The Corning filters 5840, 5860, 5874, and 5970 are heat-resisting types, and No. 5874 is recommended for the higher temperatures. No. 9863, also designated as Red Purple Corex A, will not withstand heat, and when subjected to intense 2500 to 3000Å. wave lengths there is a solarization effect or decrease in transmission. Even after solarization the ultraviolet transmission remains well above that of the other filters, and the loss of transmission can be partially restored by careful heating to about 450°C. The Corex A glass can be manufactured only in the form of flat plates.

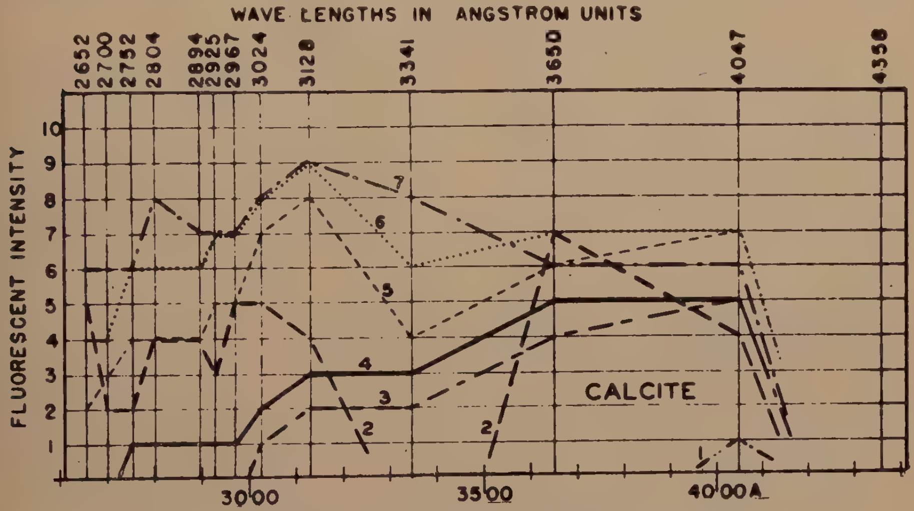

FIG. 4.—VARIATIONS IN FLUORESCENT INTENSITY OF CALCITE. Data from Smith and Parsons.[10]

| | Fluorescent Colors | |
|---|---|---|
| 1. Calcite ($CaCO_3$), Montana | red | 4047 |
| 2. Calcite, Chisos Mountains, Texas | blue<br>pink | 2652–3128<br>3650–4047 |
| 3. Calcite, Imperial County, Calif. | red | 3024–4047 |
| 4. Calcite (green), Calif. | pink<br>red | 2804–2967<br>3024–4047 |
| 5. Calcite, Franklin Furnace, N. J. | pink<br>pink-orange<br>orange<br>orange-pink | 2652–2894<br>2825<br>2867–3128<br>3341–4047 |
| 6. Calcite, Franklin Furnace, N. J. | orange-pink<br>pink<br>pink-orange<br>orange<br>pink-orange | 2652–2700<br>2752<br>2804–2894<br>2925–3341<br>3650–4047 |
| 7. Calcite, Franklin, Furnace, N. J. | pink<br>orange-pink<br>orange | 2652–2925<br>2967<br>3024–4047 |

The Jena filter, UG4, is described as highly heat resisting, and its transmission curve is very similar to Ultra 5874.

Corning and Jena filters, as shown by the transmission curves in Fig. 3, show one group most efficient for 3600 to 3700Å., and each includes a single filter efficient for the entire ultraviolet region from about 4000Å. to less than 2500Å. It is not possible with these filters to isolate the range of 2500 to 3100Å., and obviously it is not possible to isolate any limited wave-length band other than the very longest ones in or near the visible range.

## HAZARDS OF ULTRAVIOLET RADIATIONS

Ultraviolet rays at 3150Å. and longer are harmless[11]; hence all black bulbs are safe. Ultraviolet rays shorter than 3150Å. will irritate the skin or eyes, so that lengthy exposure to them should be avoided. The effect on the skin is something akin to a sunburn, although more painful. A similar sunburn of the eyes is not only painful but, according to some authorities, may result in permanent harmful effects. Makers of germicidal tubes emphasize the necessity of installation that prevents direct exposure to them.

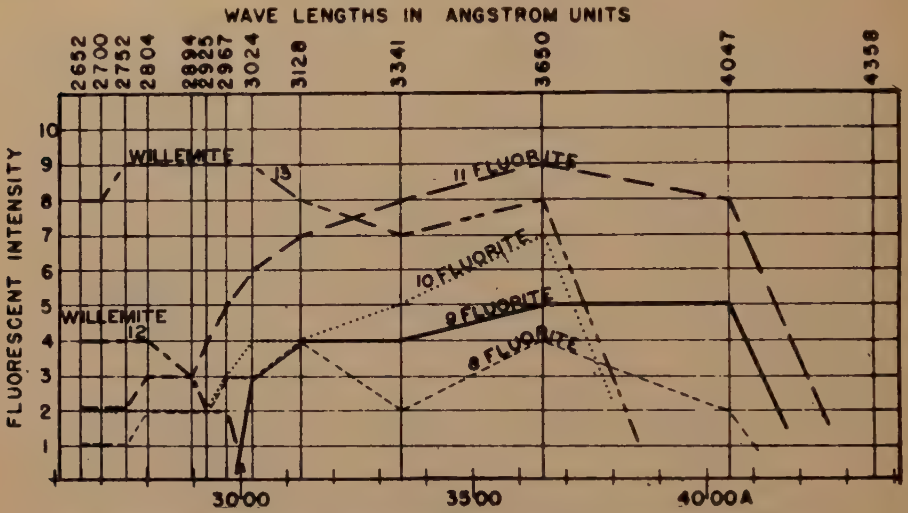

FIG. 5.—SOME VARIATIONS IN FLUORESCENT INTENSITY OF FLUORITE AND WILLEMITE. Data from Smith and Parsons.[10]

| | Fluorescent Color | |
|---|---|---|
| 8. Fluorite ($CaF_2$) (yellow), Ohio | brown-green | |
| 9. Fluorite, St. Lawrence Co., N. Y. | blue | |
| 10. Fluorite, Montana | blue | |
| 11. Fluorite (blue), England | blue-violet | 2652–2804 |
| | blue | 2894–4047 |
| 12. Willemite ($Zn_2SiO_4$), Franklin Furnace, N. J. | green | |
| 13. Willemite, Franklin Furnace, N. J. | green | |

The effect of these shorter ultraviolet rays is not well understood, and for some purposes it may be beneficial, but applications should be under expert medical advice only. Ordinarily no danger results from ultraviolet rays in observing fluorescence, first because the quantity of ultraviolet energy is small and the rays are generally directed toward specimens and not the eyes, and second the lamps are usually turned on for only a few seconds or minutes at a time. Clear or colored glass goggles or the ordinary glass spectacles will filter out all rays harmful to the eyes, and gloves may be worn to protect the hands where lengthy exposures to the shorter rays are in prospect. The writer has not heard of a single instance of harm resulting from the use of portable ultraviolet lamps.

## COMPARISON OF FLUORESCENCE WITH TWO TYPES OF ULTRAVIOLET RADIATIONS

Numerous lists of fluorescing minerals have been published, but the source of ultraviolet energy or the approximate wave lengths that may have been used is seldom given. The following partial list includes some of the more common minerals, to show the variation obtained with two sources of ultraviolet energy:

| | Purple X Globe 3200–4000Å. | Quartz Tube with Filter (Mineralight) 2500–4000Å. |
|---|---|---|
| Schroeckingerite (dakeite) ($Na_2O.3CaO.UO_3.3CO_2.SO_3.10H_2O$), Lost Creek, Wyoming.......... | Yellow | Yellow |
| Semi-opal ($SiO_2.nH_3O$), Virginia Valley, Nevada. | Canary yellow | Yellow green |
| Chalcedony ($SiO_2$), Barstow, California. | None | Light green |
| Hyalite ($SiO_2.nH_2O$), Mohave Desert, California............ | None | Green |
| Hyalite ($SiO_2n.H_2O$), Box Elder County, Utah. | None | Yellow green |
| Fluorite ($CaF_2$), Cumberland, England. | Blue | Faint blue |
| Calcite ($CaCO_3$), Franklin Furnace, N. J.................... | None | Red |
| Calcite ($CaCO_3$), Terlingua, Texas. | Flesh pink | Blue white |
| Calcite crystal, Florida.......... | Yellow | Faint yellow |
| Calcite crystal, Joplin, Missouri. | Red | None |
| Calcite many varieties.......... | None | None |
| Willemite (red) ($Zn_2SiO_4$), Franklin Furnace, N. J............. | None | Green |
| Willemite (white to green) ($Zn_2SiO_4$), Franklin Furnace....... | Green | Green |
| Scapolite (complex silicate), Quebec. | Yellow orange | Faint yellow |
| Scheelite ($CaWO_4$), Colorado, Utah, Nevada, California, Idaho, and Virginia. | None | Blue white |
| Scheelite ($CaWO_4$) (with Mo content)..................... | None | Yellow |
| Powellite ($CaMoO_4$), Colorado, Utah, Nevada, California, and Idaho. | None | Golden yellow |
| Hydrozincite ($ZnCO_32Zn(OH)_2$.. | None | White |
| Zircon ($ZrSiO_4$) (cut blue gem)... | Red | Orange |

Some minerals react about in the same way; some show only a slight difference; while scheelite and some varieties of calcite that fluoresce strongly with shorter ultraviolet do not fluoresce at all under longer ultraviolet. Many minerals show differences similar to that shown by scheelite and calcite or the fluorite from England. These differences are less easily explained than they are striking to the eye.

A detailed study in mineral fluorescence of 43 specimens has been made by Smith and Parsons[10] covering the range of 2652 to 4358Å. This work is of interest because it shows the fluorescent response to very limited bands of ultraviolet resonances. Characteristic variations in fluorescent response are shown in Figs. 4, 5, and 6, in which the rating of 5 to 6 signifies average response and the rating one is very weak fluorescence. Fig. 4 gives the erratic variations in fluorescent response shown by several calcites. Fluorite (Fig. 5) responds better to the longer wave lengths; whereas one variety of willemite (Fig. 5) is negative to the longer wave length, and one variety is not. Scheelite (Fig. 6) is negative to wave lengths over about 3100Å.

Nine minerals showed a color change from blue to pink or violet to blue—that is, from relatively short to longer wave lengths as the ultraviolet wave lengths increased, but opposed to this group four minerals showed an opposite effect. The work shows conclusively that fluorescent response depends more on the particular wave length used than on the intensity of the wave length.

The fluorescent response of zinc silicate (willemite) and calcium tungstate (scheelite) to wave lengths from 1000 to 3150Å. was studied by Beese,[2] and the results are summarized in Fig. 7. The response to wave lengths from 2250 to 3150Å. was measured with a photocell, so that the numerical value is on a different unit base than the response in the Schumann region 1000 to 2600Å., which required photographic measurement. In the overlapping region 2200 to 2600Å., the two methods used agree in showing a relatively low response at 2600Å. and a maximum response around 2300Å. The results shown in Fig. 7 show conclusively that the maximum fluorescent response for calcium tungstate is at 2700Å. and 2300 to 2350Å. The synthetic willemite shows approximately the same response as the natural mineral, and presumably this would be true for scheelite also.

The effect of a broad band of wave lengths or a continuous spectrum of ultraviolet radiations as compared with the effect of a single wave length or a very narrow band has not been studied, as far as the writer is aware. Also, the fluorescent response to a continuous ultraviolet spectrum need not be the same as the response to the banded spectrum of the mercury vapor arc. Answers to these questions require further study, but they do not change the problem of practical application of fluorescence. In the present state of our knowledge of fluorescence, it is necessary to consider the fluorescent response in terms of the source of the ultraviolet energy, including the filter that is used. With this precaution, reasonable consistency in results can be hoped for.

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[10] References are at the end of the paper.

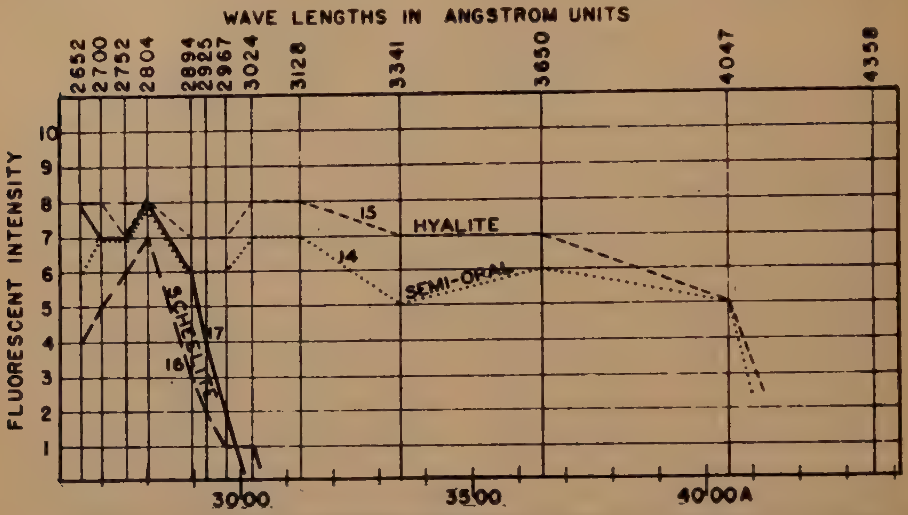

FIG. 6.—SOME VARIATIONS IN FLUORESCENT INTENSITY OF SEMI-OPAL, HYALITE, AND SCHEELITE. Data from Smith and Parsons.[10]

| | | FLUORESCENT COLOR |
|---|---|---|
| 14. | Semi-opal ($SiO_2.nH_2O$), Nevada | green |
| 15. | Hyalite ($SiO_2.nH_2O$), North Carolina | green |
| 16. | Scheelite ($CaWO_4$), Bohemia | blue |
| 17. | Scheelite, California | blue |

Studies with artificial compounds have shown that certain elements in minute quantities induce fluorescence, but equal quantities of other elements may be detrimental. For example, in zinc silicate activated with manganese, 0.001 per cent iron reduced fluorescence materially, and 0.01 per cent reduced fluorescence by 20 per cent. Natural minerals tend to respond in the same way as synthetic compounds, and since natural minerals usually contain traces of impurities variations in fluorescence are to be expected.

## FLUORESCENCE OF SCHEELITE

The use of fluorescence in mining has become very important with respect to scheelite (calcium tungstate). Powellite (calcium molybdate) fluoresces consistently; but thus far powellite has not been found in sufficient quantity to constitute an ore. The orange fluorescence of zircon (zirconium silicate) should aid in its identity in sands and thus be helpful in the exploration for zircon. Some diamonds fluoresce, but some do not, which limits the use of fluorescence in this field. The brilliant fluorescence of willemite is a decided aid in identifying this mineral, but willemite in quantity is limited to one locality, Franklin Furnace, New Jersey, where its fluorescence is utilized in checking certain mill products. The fluorescence of hydrozincite, an alteration by-product of lead-zinc deposits, offers a possibility for the useful application of fluorescence in the recognition of zinc mineralization.

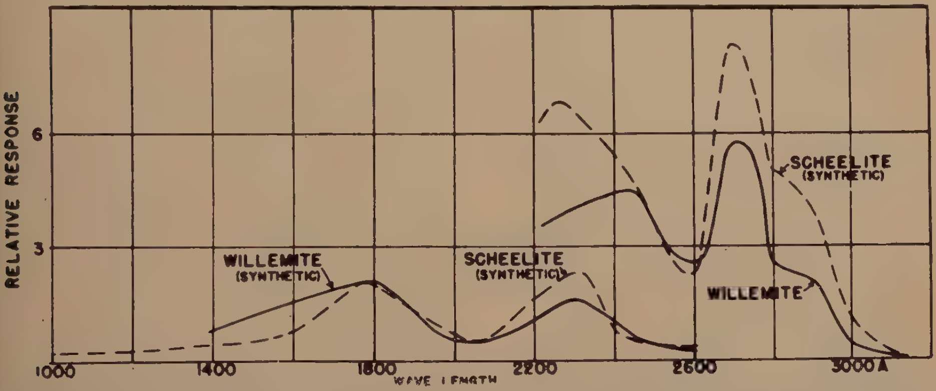

FIG. 7.—RELATIVE RESPONSE IN FLUORESCENCE OF SYNTHETIC SCHEELITE AND BOTH SYNTHETIC AND NATURAL WILLEMITE TO ULTRAVIOLET RADIATIONS IN RANGE OF 1000 TO 3150 ÅNGSTROMS. Data from Beese.[2]

However, none of these possible applications compares with the usefulness of fluorescence in the search for scheelite.

Scheelite, fortunately for the prospector, fluoresces more consistently than most minerals. Reports of varieties of scheelite that do not fluoresce have not been substantiated, nor is the claim definitely established that pure scheelite fluoresces other than blue. The blue color is pale, and its designation as blue white, or even white, can be attributed to the difficulty of accurately determining a particular shade of color. A relatively large area of scheelite will appear to have a lighter shade than small grains in a dark groundmass. The color of the scheelite itself also influences somewhat the fluorescent hue, but all in all a definite bluish cast is characteristic for most, if not all, varieties of relatively pure scheelite.

The fluorescence of scheelite has been utilized extensively for only a little more than 10 years in prospecting as well as mining this mineral. The first ultraviolet lights were expensive and somewhat crude, but the results were none the less startling. At the present time an experienced prospector would not consider exploring for scheelite without a portable ultraviolet unit; however, many assayers still get along without this convenient tool in their laboratories. There are many who profess to be able to recognize scheelite with the unaided eye, but a practical test on ordinary low-grade ores usually fails to substantiate such claims.

Fluorescence will make even small quantities of scheelite stand out conspicuously—literally a light in the dark. In addition, after the character of scheelite ore has been determined, and with experience, an operator can judge by means of fluorescence the approximate percentages of tungsten present. A word of caution is necessary in judging percentages, for ordinarily they are inaccurate, and estimates of tungsten content are usually much too high. It is essential for an operator to have an intimate knowledge of the ore and the advantages of numerous assays to check estimates. Without such a background estimates by means of fluorescence should be used with care, or, better still, avoided entirely.

Relatively few minerals fluoresce the blue to blue-white characteristic of scheelite. However, several minerals fluoresce with yellow colors similar to scheelite with small quantities of chemically contained powellite. Therefore, precautions and checks are necessary. Fluorescence is not reliable for positive identification of any mineral. It is an aid to attract attention to particular minerals which must then be identified either by chemical analysis or recognition of diagnostic physical properties. With a little experience, most of the everyday fluorescing minerals can be recognized on sight. However, as more than one hundred fairly common minerals fluoresce, it is always necessary to be prepared for the unexpected.

The following minerals, which could be confused with scheelite, are those most likely to be found in association with ore deposits:

Amorphous and hydrated silica, such as semi-opal, opal, and hyalite; fluorescence pale yellow to yellow to orange.

Calite, either coarse or fine grained. Incrustations found along joint and other crevices are particularly common. Varieties fluoresce white to bluish white.

Oil usually fluoresces, and it has a way of getting scattered around, particularly underground, so as to cause possible confusion. Fluorescence is white to bluish white.

Hydrozincite is common in oxidized parts of ore bodies. Common fluorescent colors are white or pale yellow.

Some uranium minerals are very fluorescent. Fluorescence usually is yellow to green.

Apatite as an accessory mineral in igneous rocks fluoresces orange.

Zircon as an accessory mineral in igneous rocks usually fluoresces orange.

Lichens, though not mineral, commonly fluoresce and can be confused for minerals. A few worms and lizards fluoresce.

The difficulty in recognizing scheelite arises from its inconspicuous white to gray color. However, after fluorescence has attracted attention to scheelite, further diagnostic checks as to density and hardness are relatively easy. Scheelite is twice as heavy as quartz, or the average rock, and this is evident by hefting if a large specimen is involved; and fine-grained material will respond to the ordinary technique of panning. Scheelite is moderately hard, but it will not scratch steel. Hydrated silica, opal or hyalite, in spite of the fluorescent color difference, can be mistaken for scheelite; but opal and hyalite will scratch steel, and they are only half as dense or heavy as scheelite. The blue-white fluorescing calcite is too soft and also too light to be confused with scheelite. Other occasional interfering minerals can be eliminated in the same way by taking into account their physical properties. As a final confirming check, a qualitative chemical test for tungsten or a standard analysis is recommended.

Scheelite with minor quantities of chemically combined powellite (calcium molybdate) fluoresces a distinct yellow. Powellite fluoresces a golden yellow. The contrast in color is marked when the two are seen side by side, but the color differences can be confusing under some conditions. The scheelite-powellite series occur in solid solution or mixed crystals, and the relative percentages of tungsten with respect to molybdenum can vary from 100 to 0 per cent. A knowledge of the approximate molybdenum content in a given scheelite is important, because minor quantities such as 1 or 2 per cent may entail a penalty in market price, and with only slightly higher percentages of molybdenum a scheelite may not even be marketable.

Recently a scheelite-powellite analyzer* was developed by Ralph S. Cannon, Geologist, and K. J. Murata, Chemist, of the U. S. Geological Survey, for determining the approximate percentage of molybdenum in a scheelite. The analyzer consists of a card 4½ by 6½ in., on which there are 11 fluorescing standards for a molybdenum content varying from 0 to 4.8 per cent, and one standard for 48 per cent molybdenum. The approximate molybdenum content of the scheelite mineral in the ore is determined by matching its fluorescing color with that of a standard on the card. Well-darkened surroundings are necessary for best results. The change in fluorescing color caused by 0.05 to 1.4 per cent molybdenum is quite marked; whereas under ordinary light the materials usually look identical.

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* Manufactured by Ultra-Violet Products, Inc., Los Angeles.

Differences in texture, regarded by some as indication of possible molybdenum content, are not reliable. Varieties of either soft earthy or hard crystalline scheelite may or may not have a molybdenum content. These particular physical features are very important as regards milling low-grade ores, because soft material is more difficult to separate than the hard and relatively coarse grains. Neither the physical properties nor the mode of occurrence indicate whether molybdenum is or is not present in scheelite.

It should be emphasized that fluorescence has become an important adjunct in the search for scheelite, and a number of other useful applications are indicated, but the degree of satisfaction obtained and the reliability of the results produced from fluorescence as applied to minerals will be in proportion to the care and understanding with which fluorescence is used.

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## DISCUSSION

Walter W. Bradley*—As I recall, the first portable units for ultra-violet light of practical use with scheelite were developed at the Nevada-Massachusetts Tungsten Mine at Mill City, Nevada, and were used underground. Ultra-violet lamps are also installed there in the mill, above the gravity-concentration tables, for control observations.

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* Formerly State Mineralogist of California, Division of Mines, San Francisco; now Berkeley, Calif.

# Field Comparisons of Some Magnetic Instruments, with Analysis of Superdip Performance

By H. L. James,* Member AIME

(New York Meeting, March 1947)

## Abstract

This paper presents the results and analysis of field trials of various magnetic instruments over anomalies ranging from 20 to 5000 gammas of vertical intensity.

The following instruments were used: Askania vertical magnetometer, Wolfson vertical magnetometer (model M-1), Hotchkiss Superdips (used both parallel to and normal to the magnetic meridian, with several settings of the "sigma" angle), Lake Superior dip needle, and the dipping circle for measurements of inclination.

The change in inclination over the measured anomalies ranged up to 4°; this caused pronounced changes in sensitivity of the Hotchkiss Superdip oriented parallel to the magnetic meridian. The variable sensitivity not only gives apparent displacement of crests of anomalies, particularly at high sensitivities, but also makes the data difficult or impossible to use in preparation of iso-anomaly maps. This weakness in field performance of the Superdip is shown to be eliminated by orienting the instrument normal to the magnetic meridian and measuring vertical intensity only, which in the latitude of upper Michigan is equal to 97 pct of the total intensity. The dip needle is seen to be relatively ineffectual for location of anomalies of less than 400 gammas.

## Introduction

The U.S. Geological Survey, in cooperation with the Geological Survey Division of Michigan Department of Conservation, has for the past three years been engaged in a restudy of the Iron River-Crystal Falls iron mining district of upper Michigan. The bedrock in a large part of this area is covered with glacial deposits as much as 500-ft thick, and, inasmuch as some of the rocks have moderate to high magnetic permeability, extensive magnetic mapping has been done to assist in unraveling the complicated structure of the area. Several types of instruments have been used, and the major purpose of this paper is to present comparative data on the field performances of some of these instruments so as to aid others who face similar problems elsewhere to evaluate and choose the instruments best suited for the work. The paper is directed primarily to geologists and engineers rather than to geophysicists, although some familiarity with the instruments is presumed.

The geology of the area consists of a complexly folded series of weakly metamorphosed sedimentary rocks, including the iron formation, which rests on older greenstones. Although the iron formation itself is only locally magnetic, some of the beds stratigraphically above the iron formation are rather strongly so. By tracing these hanging wall magnetic anomalies valuable information is obtained from which the structure and distribution of the iron formation can be inferred. Critical anomalies range from 25 to 6000 gammas of vertical intensity. Many anomalies can be traced in part with the common dip needle, but detection of others requires more sensitive instruments.

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Published by permission of the Director, U.S. Geological Survey. Manuscript received at the office of the Institute April 15, 1947; revised Sept. 24, 1947. Issued as TP 2293 in Mining Technology, March 1948.

* U.S. Geological Survey, Iron River, Mich.

The two test lines chosen cross typical anomalies, although the anomaly on one (line *A*) is higher than most, and the anomalies on the other (line *B*) are more closely spaced than is common.

## INSTRUMENTS

The following instruments were used in the tests: Wolfson vertical magnetometer, Askania vertical magnetometer, three Hotchkiss Superdips (used both parallel to and normal to the magnetic meridian, with several settings of the sensitivity), the common dip needle, and the Hotchkiss Superdip used as a dipping circle.

*Magnetometers*—The Wolfson magnetometer used was a Schmidt-type, temperature-compensated instrument (model M-1), with a sensitivity set at 29.0 gammas per scale division. The Askania magnetometer was not temperature compensated; it was used with a sensitivity of 17.8 gammas per scale division. No base instrument was used during the trials; the diurnal correction was obtained by periodic determinations, never more than four hours apart, at the base station. In addition to the diurnal factor, a correction of 13 gammas per degree Centigrade was applied to the Askania readings.

Both instruments were found to yield reliable and consistent results. The average error in redetermination of stations was about 8 gammas, but a large part of this error undoubtedly resulted from imperfect diurnal corrections and from slight differences in setup position over steep magnetic gradients (which in places amounted to 15 gammas per foot). The average instrumental error is probably less than 5 gammas. Aside from the error from improper diurnal correction, the largest single source of error in both instruments appears to be caused by thermal lags in some parts of the instruments when changes in temperature, particularly sudden changes, occur.

Inasmuch as the average error of station redeterminations was almost identical for both instruments, it seems likely that the sensitivity of the Askania was set somewhat greater than the optimum value for this particular instrument. There is a distinct advantage in lower sensitivity setting in that larger anomalies can be measured before use of auxiliary magnets is necessary.

*Superdips*—Three Superdips were used over the trial lines. These instruments were constructed by the Division of Field Equipment of the U.S. Geological Survey, with the permission of the patent holders. The only significant difference in their construction, as compared with that of the commercial models, is in the use of hardened chrome alloy for knife edges in two of the instruments, whereas sapphire was used in the third.

Earlier work had indicated that one of the major sources of error in Superdip work was caused by static accumulations on the nonconducting glass face of the instrument. The normal methods of dissipation of this charge (such as touching with a wet finger or breathing on glass) are not always effective; frequently there remains a residuum strong enough to affect the swing of the needle. This problem was overcome by the use of special glass windows coated with a transparent conductor; these were obtained from the Corning Glass Works. It was found that static could be removed also by use of radioactive strips placed inside the instrument; emission of alpha rays ionizes the air so as to make it conductive and thus removes the unbalanced charge from the glass. The strips were obtained by the Division of Field Equipment of the U.S. Geological Survey from the United States Radium Corporation of America.

Determinations were made with the Superdips by releasing the needle from the zero position of the dial and averaging the end points of the first forward swing, the reverse swing doubled, and the second forward swing. This method yields the approximate theoretical rest position (it is not exactly so, since the release is from a point outside of the normal arc of swing, and, furthermore, the swing is not strictly a simple harmonic motion). The instrument was set to read between 80 and 90 at the base value for the area; as will be shown later, a setting of about 75 would be better. Although the above described method has been used widely on this and other surveys, it is not necessarily recommended over the more common method of reading the end point of the first swing only. Its advantages are chiefly in reducing the error caused by faulty release and in providing a means of detecting static interference, which was revealed by abnormalities in swing. However, with static now eliminated as a cause of error, the extra time taken by the three-swing method may not warrant its use. All determinations were corrected for temperature.

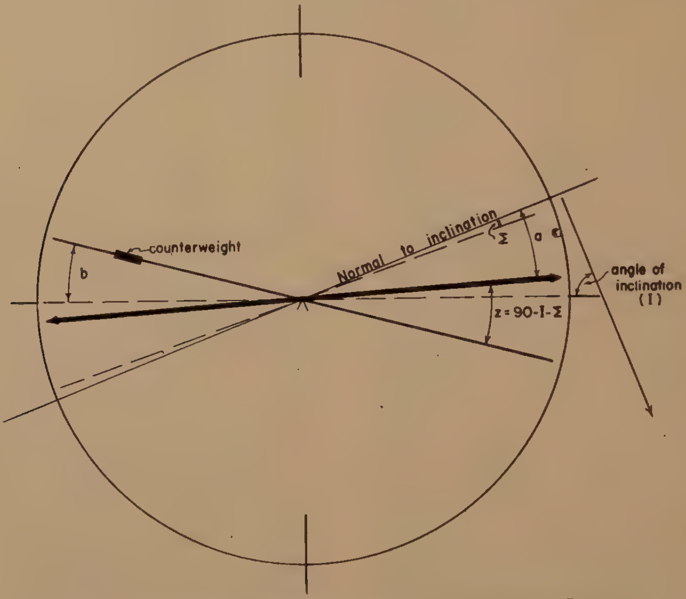

FIG 1—DIAGRAMMATIC REPRESENTATION OF HOTCHKISS SUPERDIP.

In standard field practice with the Superdip, as described by Stearn,[1] the instrument is oriented parallel to the magnetic meridian and measurements are made of the total intensity. The sensitivity of the Superdip is varied by adjusting the angle between the magnetized needle and the needle which bears the counterweight, as shown in Fig 1. This is commonly expressed as the "sigma" angle (Σ), which is the angle the magnetized needle makes with the normal to the magnetic inclination when the counterweight arm is horizontal. From Fig 1 it is seen that the angle between the magnetized needle and the counterweight arm in degrees is

$$z = (90 - I) - \Sigma$$

At any position of rest, the equilibrium condition is

$$TM \cos a = emg \cos b$$

where $a$ = angular displacement of the magnetized needle from the normal to the earth's magnetic inclination.

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[1] References are at the end of the paper.

$b$ = angular displacement of the counterweight arm from horizontal.
$T$ = total intensity of the earth's field in the plane of the magnetic meridian.
$M$ = product of the magnetic moment of the magnetized needle and its lever arm around the axis of rotation.
$e$ = lever arm of the counterweight around the axis of rotation.
$m$ = mass of the counterweight.
$g$ = force of gravity.

By construction, $b = a - \Sigma$

then, $TM \cos a = emg \cos (a - \Sigma)$

or,
$$T = \frac{emg \cos (a - \Sigma)}{M \cos a}$$
$$= \frac{K \cos (a - \Sigma),}{\cos a}$$
where $K$ is a constant.

Expanding and dividing by $\cos a$,

$$T = \frac{K (\cos a \cos \Sigma + \sin a \sin \Sigma)}{\cos a}$$
$$= K (\cos \Sigma + \tan a \sin \Sigma).$$

Differentiating,

$$\frac{dT}{da} = K \sin \Sigma \sec^2 a$$
$$= \frac{K \sin \Sigma}{\cos^2 a}$$

or, inverting, the rate of change of angular deflection with change of intensity is

$$\frac{da}{dT} = \frac{\cos^2 a}{K \sin \Sigma}$$

From this equation[a] it is readily seen that as deflections approach 90° (plus or minus) from the normal to the magnetic inclination, $\cos^2 a$ (and therefore sensitivity) approaches zero, also that sensitivity is a direct function of $\sin \Sigma$, which for small values is practically equal to the angle. In using the Superdip oriented parallel to the magnetic meridian to measure total intensity, the sigma angle—that is, sensitivity—can be varied either by mechanical changes in the angular separation of the needles or by changes in the magnetic inclination of the earth's field, since from Fig 1 it is seen that $\Sigma = 90 - I - z$. The actual measured changes in inclination over the test lines ranged up to four degrees, as shown in Fig 2 and 3. From this it is apparent that even though the physical setup of the instrument remains unchanged the actual sensitivity may be highly variable. At some stations the instrument approaches infinite sensitivity (that is, sigma approaches zero), so that the needles are in balance at any position on the circle, and no determination is possible. This was the case at stations 15 and 16 on line $A$ (Fig 2) and at station 14 on line $B$ (Fig 3), and is of common occurrence on the north side of strong anomalies. On the south side of the anomalies the sensitivity is reduced below normal value. The data are thus difficult or impossible to use in preparation of iso-anomaly maps unless inclination measurements are made at each station (which is impractical). Furthermore, because of the increased sensitivity resulting from increase in the angle of inclination on the north side of strong anomalies, crest lines are often appreciably displaced northward. As shown on Fig 2, the displacement is progressively more noticeable with increased sensitivity: at 5° sigma, the crest was displaced 100 ft; at 4°, 150 ft; and at 3°, 200 ft.

This weakness in performance of the Superdip is eliminated by orienting the instrument normal to the magnetic meridian and measuring vertical intensity ($Z$) only, which in the latitude of upper Michigan is equal to 97 pct of the total

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[a] See C. A. Heiland: Geophysical Exploration. Prentice-Hall, Inc. (1940), p. 344 for a somewhat different derivation and result.

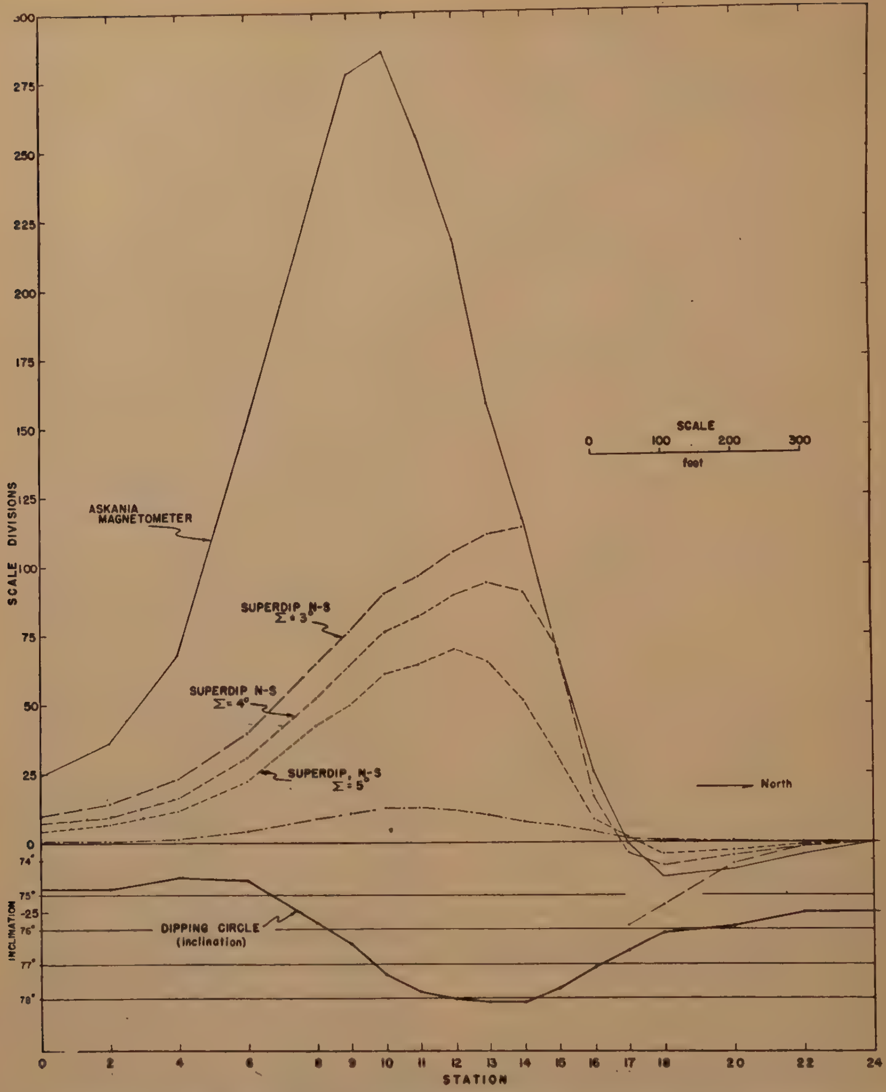

FIG 2—PROFILES ALONG LINE *A* OF DIP NEEDLE, SUPERDIP (NORTH-SOUTH ORIENTATION), AND DIPPING CIRCLE DATA. MAGNETOMETER PROFILE ALSO SHOWN. All values in scale divisions for each instrument.

intensity (*T*).[a] So far as the analysis of behavior is concerned, this merely corresponds to the unique situation where the inclination is 90°, which, with the instrument oriented parallel to the meridian, pertains only at the north pole of the earth. The relation of sensitivity to sigma angle and angular deflection remains the same as discussed above, but with the elimination of the horizontal component the effective inclination is always 90°, and an unpredictable variable is completely eliminated. The sigma angle becomes identical

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[a] This method is now used by many operators, but so far as the writer can discover, is not specifically described in the literature.

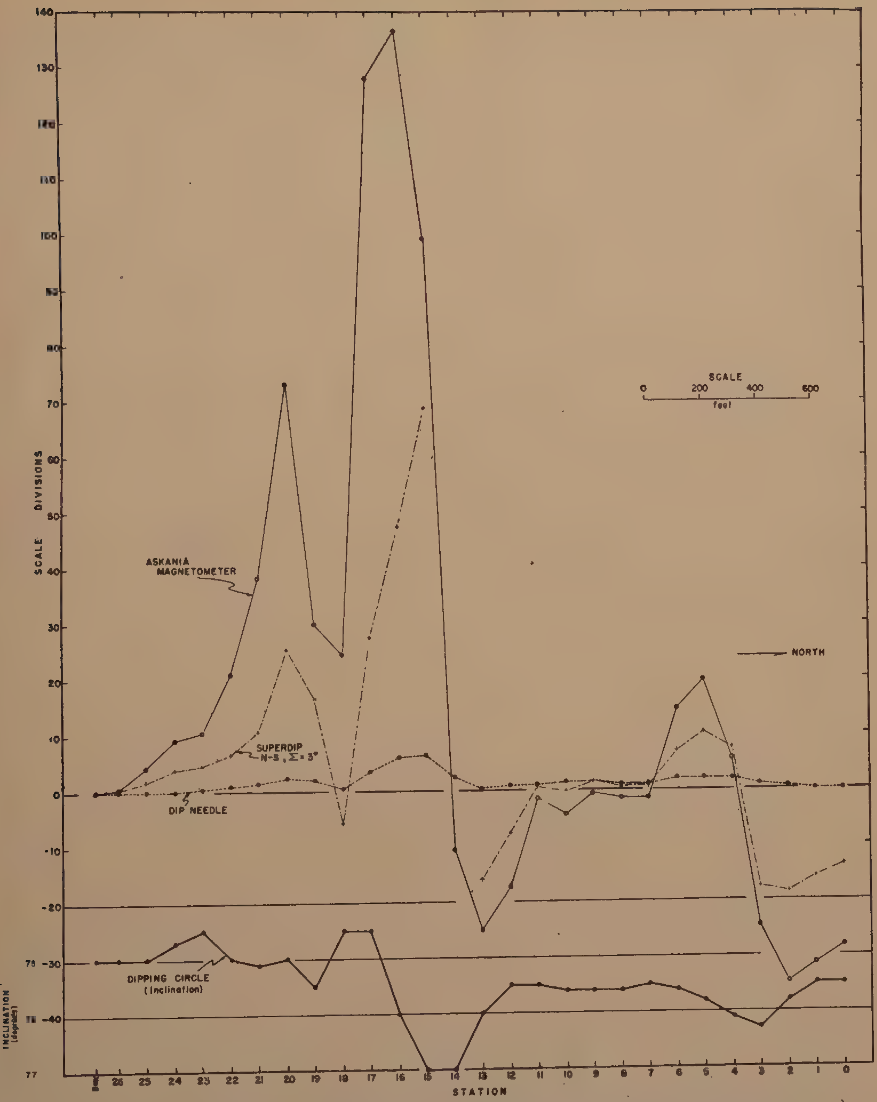

Fig 3—Profiles along line *B* of dip needle, Superdip (north-south orientation), and dipping circle data. Magnetometer profile also shown. All values in scale divisions for each instrument.

with the angular separation of the needles, and remains a constant as long as the system is physically unchanged. Comparison of profiles of determinations made with the Superdip normal to the magnetic meridian with those of the magnetometers are shown on Fig 4 and 5; the effect of lowered sensitivity at high intensities is particularly noticeable on Fig 4. Fig 5 is also of interest in that it shows the response of the Superdip to very low anomalies (less than 25 gammas), as at station 9.

Fig 6 shows the character of the sensitivity curves for three values of sigma for one instrument, as derived empirically by plotting Superdip determinations against the average gamma values obtained with the magnetometers. It should be noted that the Superdip values are of approximate theoretical rest positions, not the end points of the first swing. By this method readings can be made which are accurate to about 0.2 scale divisions. The sensitivities for this particular instrument range from 28 gammas per scale division for the straight line part of the curve for sigma = 1½° to 55 gammas per scale division for sigma = 3°. Practically, other factors, such as the needles not being balanced at their exact centers of mass, do affect sensitivity; tests of two Superdips showed that one had a sensitivity at 2° equal to other at 1½°, but for any one instrument the sensitivity will vary almost directly as the sigma angle. Several other minor factors which affect the accuracy of Superdip determinations are discussed by Stearn.[2]

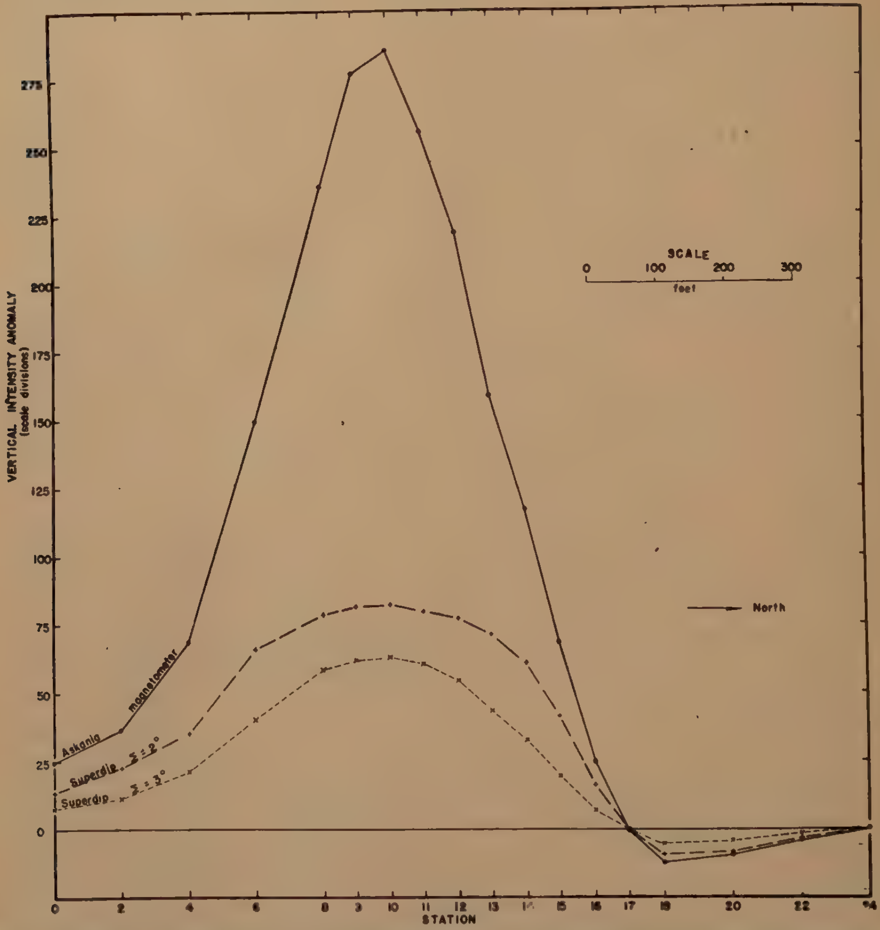

Fig 4—Profiles along line *A* of magnetometer and Superdip (east-west orientation data.

The curves shown on Fig 6 bring out clearly that in an area where most anomalies are positive, the instrument should be

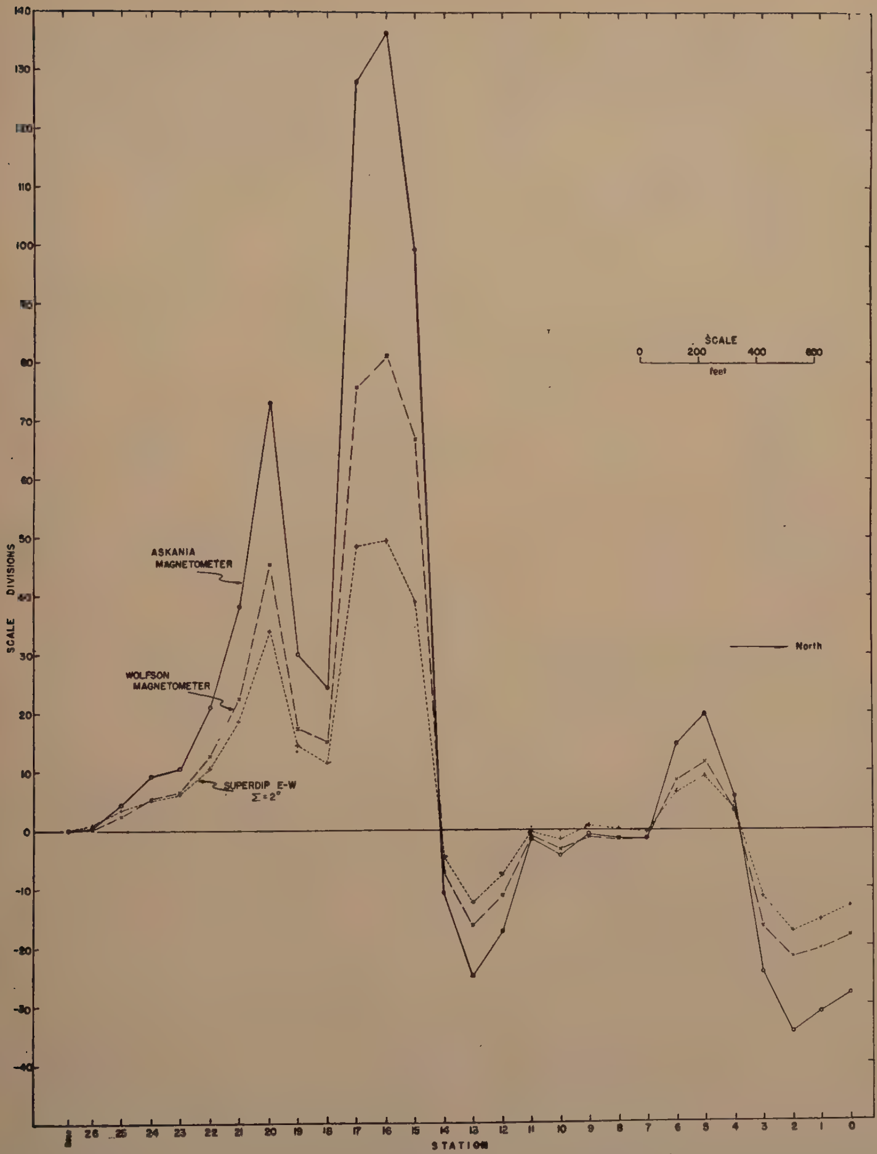

FIG 5—PROFILES ALONG LINE *B* OF MAGNETOMETER AND SUPERDIP (EAST-WEST ORIENTATION) DATA.

balanced so as to read about 15 scale divisions less than horizontal at the base value for the area, so as to take advantage of a longer section of nearly straight line (and higher) sensitivity. On Fig 7 the results of a single Superdip traverse ($\Sigma = 2°$) are plotted as a sensitivity curve against the average gamma values obtained in four magnetometer traverses over the same line.

A sigma angle of about 1½° appears to be the approximate practical limit of sensitivity settings of the instruments tested; at settings of smaller sigma angle, increased error in redetermination cancelled out the advantage of the greater sensitivity.

apparent, therefore, that the scale-division difference obtained at one temperature between two stations of different magnetic intensity will differ from that obtained at another temperature. In order to compare

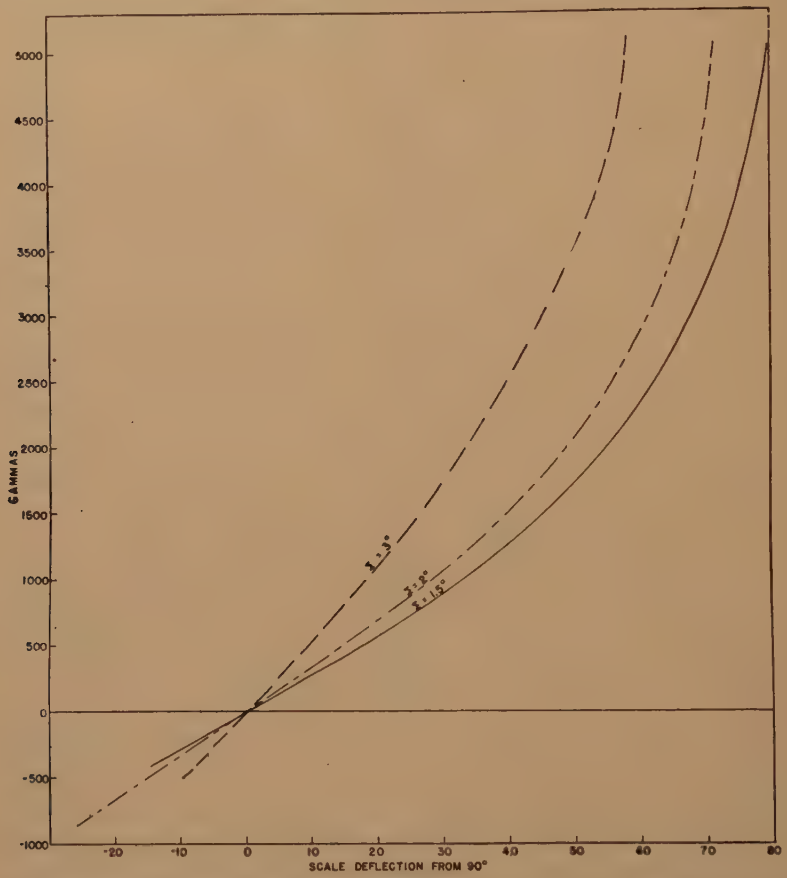

FIG 6—COMPARATIVE SENSITIVITIES OF SUPERDIP AT 1.5, 2, AND 3 DEGREE SIGMA ANGLES.

As has been shown, the sensitivity of the Superdip is not a straight-line function of intensity; it is a curve which in part is dependent upon the position of the needle with respect to the horizontal; or, more accurately, the $(90 - \Sigma/2)$ position. This gives rise to a special problem in converting the scale-division data to absolute units. The actual (uncorrected) reading at a station will vary with the temperature at which the determination was made. It is the determinations accurately, it would be necessary to establish the position of the actual (uncorrected) readings with respect to the $(90 - \Sigma/2)$ position so that different segments of the sensitivity curve would be used. The difference in value of an anomaly between a determination made in the summer compared with one made in the winter can easily amount to several scale divisions.

*Dip Needle*—The dip needle used for comparison was of the so-called Lake

Superior model. Readings were made in the usual fashion by the observer facing west and holding the instrument in the magnetic meridian. The determination is the average of one forward and one reverse swing, and is accurate to about 0.5 scale divisions. Like the Superdip oriented in the magnetic meridian, it also is affected by the change in inclination, so that crests of strong anomalies are displaced northward, as shown in Fig 2, but this displacement is very slight compared with that shown by the Superdip.

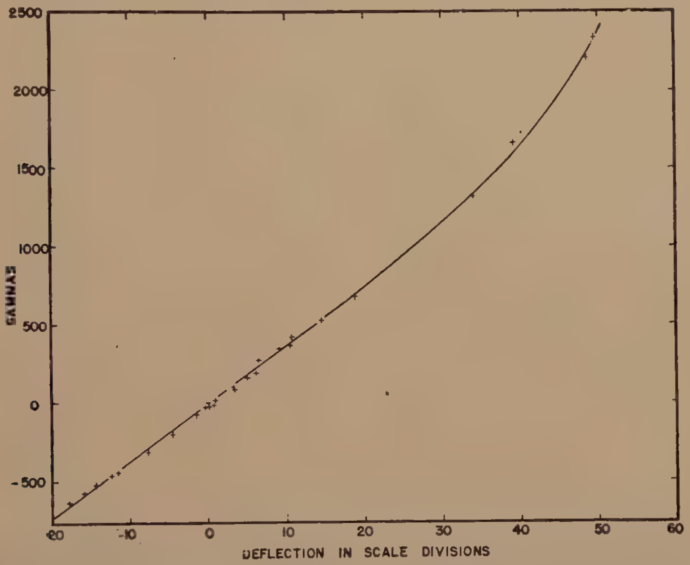

FIG 7—DATA FROM TRAVERSE WITH SUPERDIP IN EAST-WEST ORIENTATION PLOTTED AS SENSITIVITY CURVE.

*Dipping Circle*—The Hotchkiss Superdip was used as a dipping circle for measurements of inclination by removing the counterweight and placing the needles in a parallel position. This rarely can be done exactly, so that determinations have to be corrected to the base station value. Determinations were made by averaging the end points of one forward and one reverse swing; with great care this can be done with an accuracy of about 0.1° to 0.2°.

*Speed of Operation*—There is no significant difference in speed of operation between the various tripod-mounted instruments. All require essentially the same operation, namely, setting up, leveling, and orientation of the tripod, placing on of the instrument, and reading. All instruments, except the Wolfson magnetometer, were operated by a single man without an assistant; the time per station varied from 2½ to 5 minutes. In the past it was found that more stations could be occupied per average day with the magnetometer than with the Superdip, because static accumulations in the latter instrument at times made it difficult to obtain consistent readings. As mentioned earlier, this difficulty has now been eliminated by use of conducting glass or radioactive foils, so that the rate of operation should be about the same as that for the magnetometer. However, the tripod orientation for the Superdip operated in the east-west position must be made with much greater precision than for the magnetometer. With the magnetometers errors in orientation of as much as 10° have only slight effect on the average determination, whereas with the Superdip, half a degree of misorientation is sufficient to give an appreciable error in determination because of the much wider amplitude of swing.

### Appraisal of Results

Comparison of the profiles of intensity measurements made with the various instruments shows the following:

1. The best results were obtained with the magnetometers, both Wolfson and Askania. These instruments are reliable both for indicating very weak anomalies (5 gammas or less), and for yielding consistent values in absolute units (gammas). The latter point is of great importance when the data are to be used in preparation of detailed iso-anomaly (magnetic contour) maps, particularly when a large area is to be surveyed and it is necessary to tie in the work of more than one instrument.

2. The Superdip oriented parallel to the magnetic meridian, although capable of recording weak anomalies, is of variable and unpredictable sensitivity, and the data are of limited value only.

3. The Superdip oriented normal to the magnetic meridian is an effective instrument capable of recording weak anomalies, but, probably because of gradual shifting of the points of contact of the axle on the knife edges, it is somewhat less trustworthy for absolute values. Station-to-station accuracy is generally much better than line-to-line accuracy. Furthermore, as has been discussed earlier, the value of an anomaly in scale divisions is dependent to some extent upon the temperature at which the determinations were made. The data are most suitably studied as profiles, although iso-anomaly maps are satisfactory except where the gradients are very low. The Superdip thus oriented is admirably suited for work in areas where anomalies have a wide range of value. In a sense, the instrument adjusts itself to the size of the anomaly to be measured; readings can be made at any intensity without recourse to auxiliary magnets, and yet the instrument retains high sensitivity over weak anomalies.

4. The common dip needle, because of its much greater speed of operation and convenience of carrying, remains a very useful instrument for determination of large anomalies, provided that it is remembered that crests of anomalies may be displaced somewhat northward of the true intensity crest.

In general, the instruments can be fitted to the range of anomalies to be measured approximately as follows:

| Value of Critical Anomalies, Gammas | Instrument | Sensitivity, Gammas per Scale Division |
|---|---|---|
| 5 to 3,000 | Magnetometer | 15 to 30 |
| 75 to 10,000 | Superdip (oriented east-west) | 30 to 60 |
| Above 600 | Dip needle | 250 to 450 |

### Acknowledgments

The writer wishes to thank Mr. K. L. Wier and Mr. J. V. N. Dorr, 2nd, of the U.S. Geological Survey, for assistance in the field work. Many of the ideas here presented were developed in discussions and correspondence with other members of the Geological Survey, including Mr. L. E. Smith, Mr. L. D. Clark, and personnel of the Division of Field Equipment. Thanks are also due to Mr. H. R. Joesting of the Geological Survey for critical reading of the manuscript.

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# A Technique for Photographing Difficult Subjects through a Petrographic Microscope

By Donald W. Scott,* Member A.I.M.E.

(New York Meeting, March 1947)

Generally speaking, there is nothing very difficult about taking good micrographs of photogenic thin sections or grains with a petrographic microscope-camera setup. However, sometimes it is desired to show some detail that cannot be illustrated satisfactorily by the usual micrographs taken with parallel polarized light, with crossed nicols, or even with two micrographs. A simple procedure giving good results has been used at Battelle Memorial Institute for subjects falling into this classification, and it is believed to be of interest, especially to research people who must sell a point to executives through the medium of well-illustrated reports.

The following problem illustrates a typical application: It is desired to photograph a thin section of an iron ore to illustrate the character and association of the iron oxides and the individual quartz crystals. A magnification of 75 dia. is acceptable.

A micrograph of the field in question (Fig. 1*a*) was taken with parallel polarized light. Due precaution was taken to provide critical illumination, correct exposure, a sharp focus, a good darkroom technique, and other essential conditions. It illustrates correctly the iron oxides, but it fails to show the individual quartz crystals. A micrograph of the identical field (Fig. 1*b*) was taken under the same precise conditions, but with crossed nicols. It gives a better idea of the character of the individual quartz crystals but is likely to be misinterpreted unless accompanied by a lengthy descriptive note. This micrograph suggests that the centrally located, large black area is composed entirely of iron oxides. That this is an incorrect interpretation is evident by reexamining the micrograph of Fig. 1*a*. The correct interpretation is that the dark area contains quartz crystals oriented with their optical directions parallel to, or nearly parallel to, the vibration directions of the polarizer and analyzer of the microscope. Thus, the executive, or anyone who had not done the work personally, would get no more than a fair interpretation, even if two micrographs were included; and in the interests of both brevity and clarity, one micrograph usually is more desirable. Exactly how many quartz crystals occur in this region cannot be told from the two micrographs, and a printing of the two negatives in register would likewise not show the detail desired. Thus, it is evident that some subjects cannot be satisfactorily illustrated by the usual procedures.

In photographing Fig. 1*c* the light was subdued and, during the entire exposure of 75 sec., the analyzer nicol was rotated uniformly in its mounting from a crossed position to 45° uncrossed position to introduce the desired amount of contrast in the various individual quartz crystals. The true reproduction of the character and association of the iron oxides, and the additional structure and detail in the quartz areas, is apparent. It might be

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Manuscript received at the office of the Institute May 3, 1946. Issued as TP 2092 in Mining Technology, November 1946.

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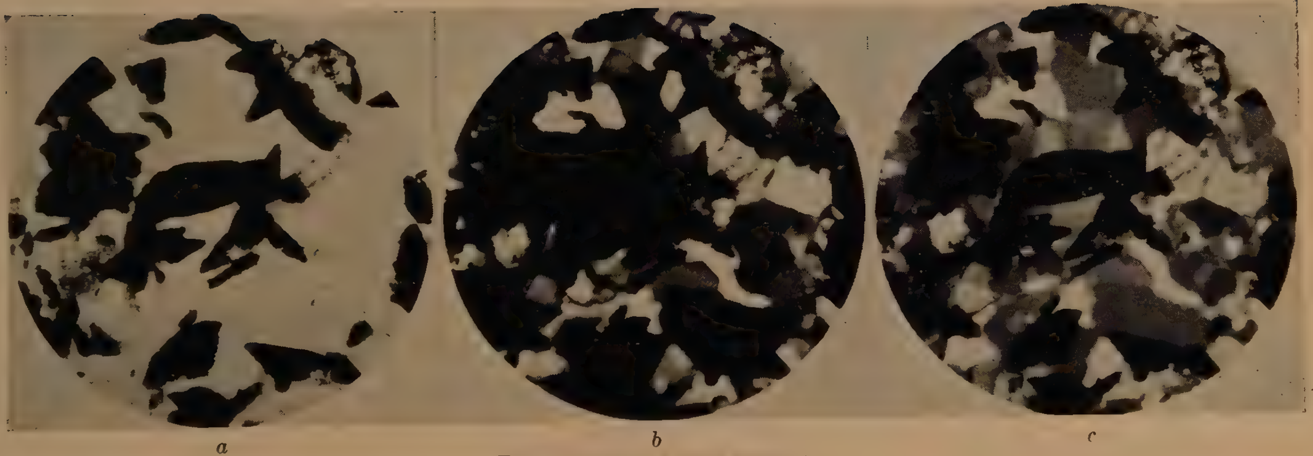

FIG. 1.—IRON-ORE THIN SECTION.

*a.* 40-second exposure, parallel polarized light. Black is iron oxide, light is quartz.
*b.* 60-second exposure, crossed nicols.
*c.* 75-second exposure, throughout which the analyzer nicol was rotated uniformly from crossed to 45° uncrossed position.
× 75, Ortho X film, D-61A developer.

argued that the same thing could be accomplished without rotation by just slightly uncrossing the nicols. This is not true because anisotropic crystals oriented so that the light travels parallel to an optic axis (the *C* axis for quartz) will be dark at the small uncrossed angle required to give contrast in the grays and blacks, and will appear similar to iron oxides. This scheme will give suitable results for some subjects if the selenite plate is inserted and panchromatic film is used. The use of color film and prints is desirable, but not yet practicable for routine work.

A suitable setup for the photographic procedure described is shown in Fig. 2. The rotation of the analyzer nicol does not disturb the field being photographed other than to continuously change the amount of light transmitted by the various individual crystals in proportion to the geometric relation between their optical directions and the ever-changing vibration direction of the analyzer. Everyone familiar with a petrographic microscope is aware of the quadruple change of color of anisotropic crystals under crossed nicols when the stage is rotated through 360°, and a result similar to that described could be obtained by synchronizing the camera with the stage and then rotating the stage during the exposure. In view of the simplicity of rotating the analyzer, the preference for this method is obvious. The amount of rotation of the analyzer, or contrast required, differs for various subjects, but is easily determined after a little experience. If the procedure is carried out properly, none of the anisotropic crystals, quartz in the illustration used, will be black, and a true presentation will result in a single micrograph.

## DISCUSSION

**Horace Winchell***—Dr. Scott's suggestion of continuously oscillating the analyzer of a petrographic microscope during a photographic exposure, to differentiate grains in a heterogeneous thin section, is an interesting and effective means for obtaining results in a difficult situation. Similar results can be obtained by observation in elliptically or circularly polarized light. This alternative seems at least as easy to apply and may sometimes be superior optically; it will attain most of the same results in practice. Illumination with circularly polarized instead of plane polarized light is obtained by placing a quarter-wave plate at 45° between the polarizer and the specimen. A second quarter-wave plate is then placed in the accessory slot between the specimen and the analyzer, oriented at 90° to the first quarter-wave plate and 45° to the analyzer. Its retardation will thus compensate that of the lower quarter-wave plate so that darkness results when there is no specimen on the stage. Elliptically polarized light is obtained if the vibration axes of lower quarter-wave plate are not oriented at 45° to that of the polarizer. Some light is then transmitted by the system when nothing is present on the microscope stage. Such a setup is used when it is desired to distinguish between opaque and isotropic

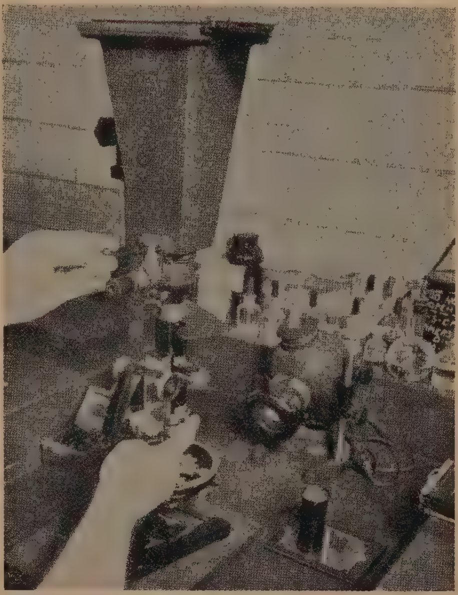

FIG. 2.—SUITABLE SETUP FOR PHOTOGRAPHIC PROCEDURE.

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* Assistant Professor of Mineralogy, Yale University, New Haven, Conn.

grains (including sections normal to optic axes of anisotropic crystals).

Photographs taken through the circular-polariscopic microscope just described are strictly analogous to those that would be obtained if, as suggested by Scott, the specimen and the camera plate could be rotated synchronously, with stationary nicols crossed or nearly crossed; or if the specimen and camera were held still and the crossed nicols were rotated synchronously. The effect of continuous rotation during a long photographic exposure is the same as that of substitution of circularly polarized for plane polarized light. Both techniques eliminate variations of grain brightness due to the four extinctions that occur on rotation in plane polarized light between crossed nicols, and preserve the variations of grain brightness due to variations in apparent birefringence resulting from different grain orientations.

Certain mechanical and optical advantages might be obtained with the circular polariscope that could not be got by Scott's method of rotating the analyzer of a plane polariscope during a photographic exposure. The principal advantage would seem to be that the resulting image may be previewed directly on the ground glass before photographing it, thus eliminating any need for special experience or practice in judging the amount and speed of rotation to be used; moreover, the effect desired can be set up and photographed several times at varying exposures with no uncertainty resulting from possible variations in speed of rotation of the analyzer, and so on.

Minor advantages that might be important under certain conditions would be the elimination of a slight shift of the image during rotation of the analyzer, noticeable in certain microscopes; and elimination of the need for a microscope fitted with a rotatable analyzer. These minor disadvantages of the rotation technique could be eliminated by rotating the polarizer instead of the analyzer, however, leaving only the one major difference cited above.

# Diamond-drill Sludge Sampling and Appraisal of a Weathered Ilmenite Ore Body, Piney River, Virginia

By D. M. Davidson,* Member A.I.M.E.

(Chicago Meeting, February 1946)

The drill sampling and evaluation of the Piney River ilmenite property was carried out during the early part of 1944 by E. J. Longyear Co., using the company's standard core-drill outfits to complete 44 holes to an average depth of 110 ft., from which were collected 1100 samples.

The Piney River ilmenite-apatite lode is 25 miles north of Lynchburg, in Nelson and Amherst Counties, Virginia (Fig. 1). Its elevation is about 900 ft. above sea level and it lies in the old pre-Cambrian complex of crystalline rocks that make up the Piedmont physiographic province lying east of the Blue Ridge Mountains. The terrain is gently rolling and well watered and is drained by the Piney and Tye Rivers, which are tributary to the James.

## Geology

The Piney River ore body comprises the upper, weathered part of a tabular basic rock, which occurs in a large area of granite-gneiss. The section containing profitable ore is about 3000 ft. long. It is separated naturally by thinning, into northeastern and southwestern segments with widths of 300 and 100 ft. respectively. The dip averages 45° southeast but steepens locally (Figs. 2 and 3).

The unaltered rock is an even, granular mixture of feldspar, ilmenite, and apatite, to which Watson[1] gave the name Nelsonite. The decomposed ore body, which is now being exploited by open cut, is a yellowish clay material in which black ilmenite and glassy apatite are the only conspicuous minerals. An economically important structural feature of the deposit is a banding of minerals coinciding with the dip and strike of the dike. This is more pronounced in the footwall of the northeastern part of the intrusive, where there is also a higher than average concentration of ilmenite.

According to Watson,[1] the $TiO_2$ content of the Nelsonites of this region ranges from 10 to 42 per cent, while $P_2O_5$ varies from 4 to 15 per cent. It is inexpedient at present to disclose exact metal content of the Piney River ore body; suffice it to say that the average tenor lies well within the extremes noted by Watson.

## Sampling Methods and Procedure

During a preliminary inspection of the ore body, Longyear engineers surmised that because of its soft, clayey character, drill sampling would yield little core, therefore they made all possible preparations to procure accurate samples of the drill cuttings, or sludge.

The first requisite in obtaining representative drill cuttings is to secure their full recovery. Only under certain conditions, and by employing a most careful and rigid sampling technique, is this possible. There must be no loss of the water circulating in the hole, and the velocity

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Manuscript received at the office of the Institute Oct. 14, 1944; revised April 30, 1945. Issued as TP 1951 in Mining Technology, January 1946.

* Manager of Exploration and Chief Geologist, E. J. Longyear Co., Minneapolis, Minnesota.

[1] T. L. Watson and S. Taber: Virginia Geol. Survey *Bull.* No. III-A (1913).

and quantity of drill water must be sufficient to lift to the surface the entire amount of material drilled.

At Piney River, there was enough clay in the ore to render it impervious throughout 5-ft. runs, with a resultant full water return. In every run, water recovery was checked by measuring and comparing the amounts pumped into and returned from the hole.

Velocities and volumes were also under constant tests. In these relatively shallow holes, velocity was measured by releasing various blue pigments or rouge from a vial at the bottom of the rods and with a stop watch timing the release and the first appearance of the colors on surface. Maintaining a pump pressure of 15 to 20 lb. in holes cased with BX ($2\frac{3}{8}$ in. i.d.) casing and drilled with B-rods ($1\frac{29}{32}$ in. o.d.) caused velocity in the returning water current of approximately 18 in. per sec. Repeated tests showed that, provided sufficient time was allowed for lag—3 to 5 min. when holes reached a depth of 100 ft.—this rate was ample to raise even the heaviest and largest particles to surface.

Nevertheless, assurance on these points by no means eliminates all the hazards attending sludge sampling. There is the ever constant threat during drilling, or cleaning out after a run, of caving into the hole, thus destroying the integrity of the cuttings as samples.

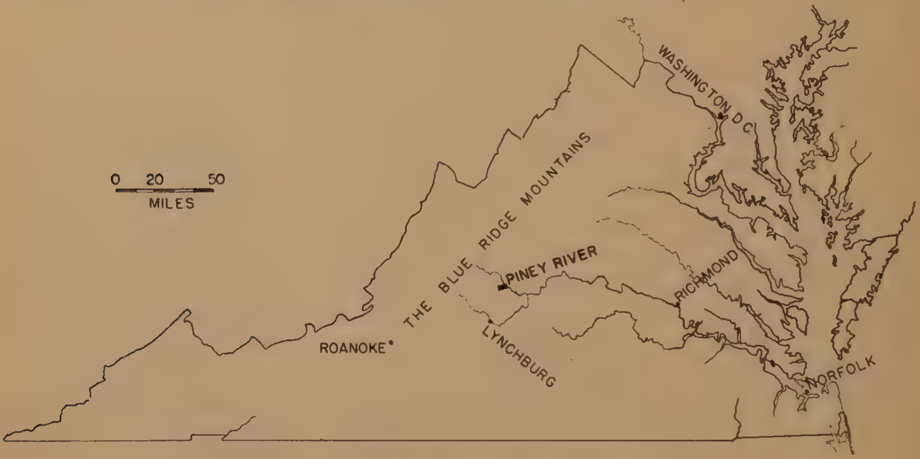

Fig. 1.—Location of Piney River, Virginia.

At Piney River, any appreciable collapse of material in the walls was prevented by driving the casing to the bottom of the hole after each run. Practically all of the sludges weighed more than the calculated theoretical dry weight for BX size, but after allowing for the varying moisture content, the overweights of the dry samples were fairly consistent—about 10 per cent. This can be reasonably interpreted to mean a uniform enlargement throughout the drill run resulting from abrasion, water erosion, and the unavoidable but minor vibration of the core barrel and rods in drilling this soft ground.

The procedure that resulted in the recovery of satisfactory sludge samples was as follows:

The overburden was penetrated by fishtail bit. When ore was reached, 3-in. stand pipe was placed and the hole was washed clean. Holes were drilled BX size with standard mechanically set bortz bits and B-rods, which, better than the smaller sizes, reduced vibration and danger of caving. Pump pressure was maintained at 15 to 20 lb., and velocity and volume of drill water were constantly checked.

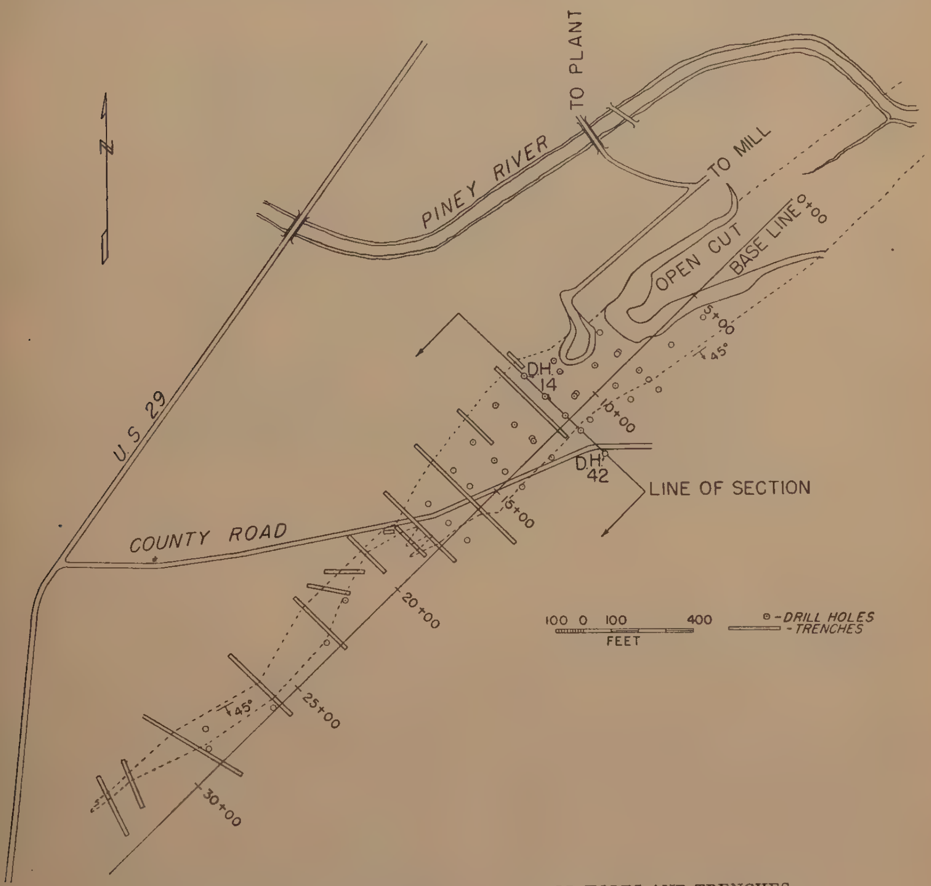

FIG. 2.—PLAN OF ILMENITE ORE BODY, DRILL HOLES AND TRENCHES.

The amount of water used in a 5-ft. run averaged 125 gal. Velocity varied from 18 to 24 in. per sec. Casing was driven to bottom of hole after each 5-ft. sample run, and the hole was washed clean before the next sample was taken. All return water with cuttings was fed into a battery of 20-gal. tubs, where solids settled satisfactorily in about 30 min. After the liquid had been siphoned off the sludge was poured into a filter (Fig. 4) where up to 90 lb. air pressure was introduced. About 20 min. sufficed to filter enough water through the canvas to leave an easily handled cake, which was removed with care (Fig. 5), weighed moist, and sent for assay. Experienced engineers were in constant supervision of the entire sampling procedure.

Despite the painstaking technique and careful checking employed to ensure accuracy in the sludges, the cost was moderate. For the sampling, it averaged about one dollar per foot, or about one third that of the drilling.

## CONFIRMATION OF SAMPLING RESULTS

In view of the general disparagement of the reliability of sludge samples, it was deemed advisable to put to the test the results of this sampling, careful though it was. Two means for this were adopted, from both of which there ensued satisfactory confirmations.

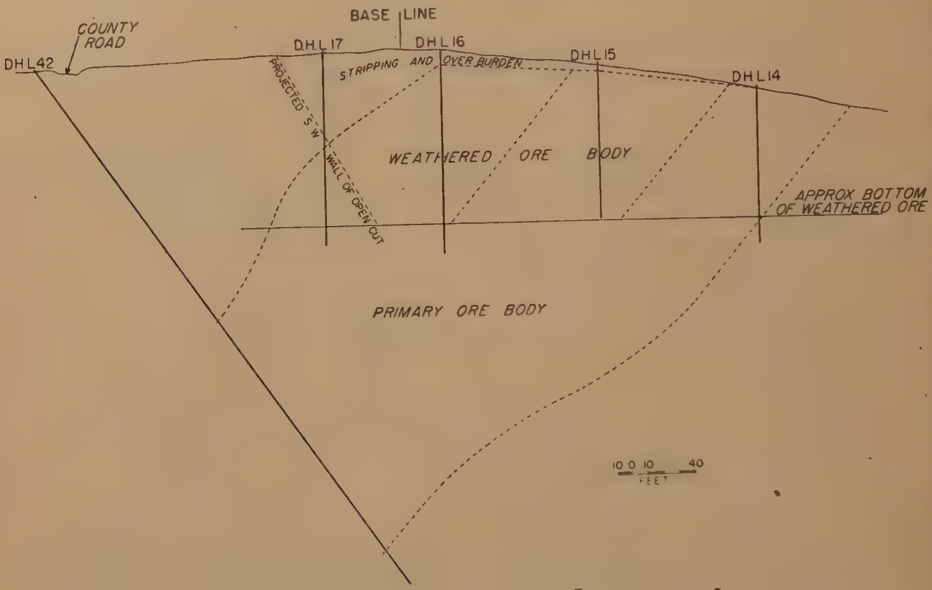

FIG. 3.—SECTION THROUGH DRILL HOLES L14 THROUGH L42.

The first was obtained by drilling duplicate check bore holes. For this purpose drill hole L4A was sunk within a few feet of L4, and drill hole L20A beside L20, and with these the following checks were obtained:

| DRILL HOLE | FEET DRILLED | $TiO_2$ PER CENT | $P_2O_5$ PER CENT |
|---|---|---|---|
| L4 | 92 | 18.2 | 6.7 |
| L4A | 92 | 18.8 | 6.4 |
| L20 | 105.5 | 11.1 | 6.0 |
| L20A | 105.5 | 11.2 | 5.9 |

Second, a comparison of the metal content of the drill-hole samples with a corresponding portion of the mined ore served as convincing evidence of the integrity of the drill samples, inasmuch as the averages differed by only 0.5 per cent $TiO_2$.

### TONNAGE AND GRADE ESTIMATE

Weight of ore in place was determined from 50 samples from trenches, the open cut, and drill cores. These also provided moisture tests. Dry weight averaged 148.15 lb. per cu. ft., or 13.5 cu. ft. to the short ton. Moisture ran 11.5 per cent.

With the lode dipping at 45° and having an average depth of 100 ft., vertical drill holes spaced at 100-ft. intervals across the strike completely cross-sectioned the wide parts of the ore body. However, in the narrower segment one 50° hole satisfactorily cross-sectioned the entire lode above the 100-ft. level. Lines of drill holes that sectioned the main part of the ore body were spaced along the strike at 100-ft. intervals. This distance was increased to 300 ft. in the south ore body, where trenches showed great consistency in the metal content of the ore.

In calculating drill-hole averages from individual samples, standard conservative engineering practice was followed. Samples of the same length were averaged arithmetically; where lengths varied, the averages were derived by multiplying length times metal content and dividing by total length.

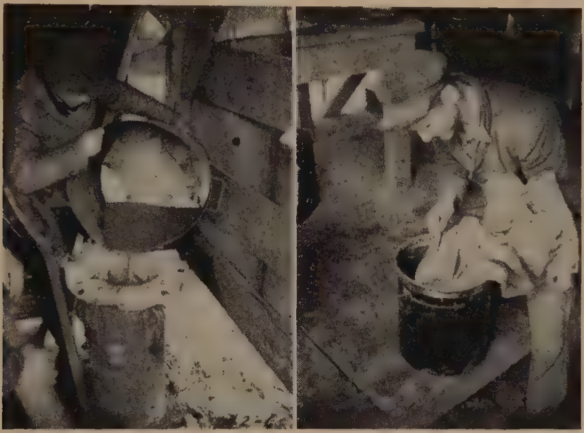

FIG. 4. FIG. 5.

FIG. 4.—SLUDGE FILTER.
FIG. 5.—REMOVING SLUDGE CAKE FROM FILTER.

In estimating the average value of the end area for drill-hole sections, due consideration was given to segregation of ilmenite mineral in bands that parallel the dip and strike in this deposit. For correct evaluation, the area of influence for each drill hole is an approximate parallelogram whose top and base are the upper and lower limits of the ore as revealed by the drill hole and whose sides are inclined parallel to the dip, the midpoint of the sides being equidistant from adjacent bore holes (Fig. 3). The average value for a section was calculated by combining the average mineral contents of the drill holes weighted according to the square feet in the encompassing parallelogram.

The ore blocks were delimited into rhombohedrons whose ends were midway between drill sections. Final tonnage was totaled from the sum of these blocks, while the grade was computed from block averages weighted according to tonnage.

## CONCLUSIONS

Drill sludges can be valid samples when taken under conditions that ensure the recovery of all the cuttings. These conditions obtain if: (1) all the drill water is returned, (2) the circulating water has sufficient velocity and volume to raise all of the particles to surface, and (3) if contamination of the sample through caving is avoided. In the Piney River drilling, these conditions were maintained throughout, with the result that confirmatory checks established the validity of the samples beyond reasonable doubt.

## ACKNOWLEDGMENTS

The writer gratefully acknowledges the courtesies and assistance rendered by the Plant Manager at Piney River, Mr. A. B. Hettrick, and his staff during the work, and is indebted to the American Cyanamid Co. and the Virginia Chemical Co. for permission to publish this paper. Much of the credit for the success of the investigation is due to the untiring efforts of J. E. Morris, engineer in charge of sampling.

# Deviation of Diamond-drill Holes in the Metaline District, Washington

By Edward Sampson,* Member A.I.M.E., and Allan B. Griggs†

(Chicago Meeting, February 1946)

## Abstract

Surveying of many holes drilled by the U. S. Bureau of Mines in the Metaline district has shown surprising deviations. The holes start in the Ledbetter slate (Ordovician) and pass into the underlying gently dipping Metaline limestone (Cambrian) at varying depths, locally in excess of 1000 ft. All holes start vertical, but one had deflected as much as 35° at a depth of only 300 ft. One hole was found to be deflected 70° at a depth of 1000 ft. These deviations were not indicated to the drillers by any action of the rods or machines.

Analysis of the survey data in the light of the observed geology indicates that the holes deviate most in the Ledbetter slate and scarcely at all in the Metaline limestone. The upper part of the "limestone" in most holes is a massive rock strongly altered by hydrothermal action and essentially homogeneous. The slate is only obscurely bedded but has a cleavage, varying from slight to strong, which is unrelated to bedding. Drill holes have deflected to become nearly perpendicular to the cleavage.

Several plans and cross sections are submitted showing the behavior of various holes.

## Introduction

The Metaline district of northeastern Washington has received much attention recently and has promise of becoming even more important as a source of zinc than at present. Exploration has been conducted largely by deep diamond drilling, and in the course of this work it was found that holes became seriously deflected. This paper deals with the nature and causes of these deflections.

The district lies along the Pend Oreille River in the northeastern corner of Washington, about 100 miles north of Spokane, 9 miles from British Columbia, and 15 miles from Idaho. Three mines are now in operation and for a number of years have furnished a considerable production of zinc and lead. The ore contains zinc and lead in a ratio of about 3½ to 1, and occurs in the upper part of the Metaline limestone of Cambrian age, the upper member of which, where unaltered, is a rather pure limestone. This is overlain by Ledbetter slate of Ordovician age. Ore has formed by replacement of limestone, commonly 100 to 200 ft. below the slate, in areas of gentle dip, resulting in extensive tabular ore bodies. In and near ore, limestone has been intensely altered to granular dolomite and fine-grained quartz, which, where it predominates, forms "jasperoid."

As part of their war program, the Geological Survey and Bureau of Mines of the U. S. Department of Interior have been cooperating in an investigation of the district. The exploration by the Bureau of Mines in 1943 and 1944 was conducted largely by deep diamond drilling in which as much as 1000 ft. or more of Ledbetter slate was penetrated before the ore-bearing Metaline limestone was reached. During the course of this drilling it was found that holes were seriously deflected while traversing the Ledbetter slate.

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Permission for publication granted by the Director, U. S. Geological Survey. Manuscript received at the office of the Institute March 28, 1945. Listed for New York Meeting, February 1945, which was canceled. Issued as TP 1825 in Mining Technology, January 1946.

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† Associate Geologist, U. S. Geological Survey.

## Recent Drilling

In recent years the mining companies found that the ore extended for long distances from the outcrop under a thick cover of slate. It has been in further search for such ore that the Bureau of Mines has concentrated most of its work. Since starting in January 1943, the Bureau has drilled 38 holes, many of which were more than 1000 ft. deep; the deepest being 2025 ft. During this period the mining companies, particularly the Pend Oreille Mining and Milling Co., have also done a large amount of deep drilling. The present paper is concerned with the holes drilled by the Bureau of Mines on the property

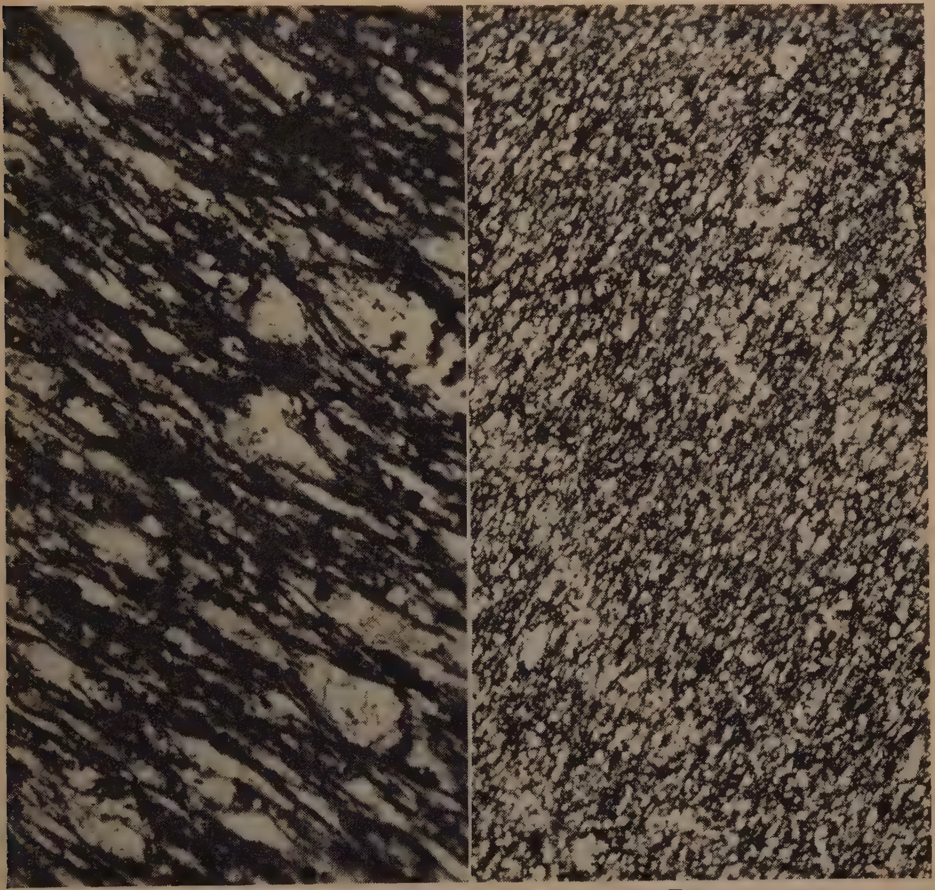

Fig. 1. Fig. 2.

Figs. 1 and 2.—Micrographs of bedding and cleavage in Ledbetter slate.

Fig. 1. From core at depth 803 ft., hole B. Smaller clear areas, quartz and some calcite; larger clear areas mostly calcite. Section cut perpendicular to cleavage and nearly perpendicular to bedding. Cleavage indicated by elongation of grains; bedding indicated by narrow bands, which are rich in calcite; and lie at 65° to the cleavage. × 53.

Fig. 2. From core at depth 102 ft., hole B. Clear areas quartz and minor calcite. Section cut perpendicular to cleavage and nearly parallel to bedding. Bedding parallel to long dimension of photograph; cleavage parallel to elongation of grains. Black interstitial material is probably a clay mineral blackened by carbon. × 200.

Micrographs by Kenneth E. Lohman.

of the Pend Oreille Mining and Milling Co. lying east of the Pend Oreille River, known as the North Pend Oreille area. The writers are indebted to this company for permission to disclose the data here presented, particularly to Mr. C. A. R. Lambly, Manager, with whom many problems of the work of the Geological Survey in the Metaline district have been discussed and whose unfailing cooperation and assistance it is a pleasure to record.

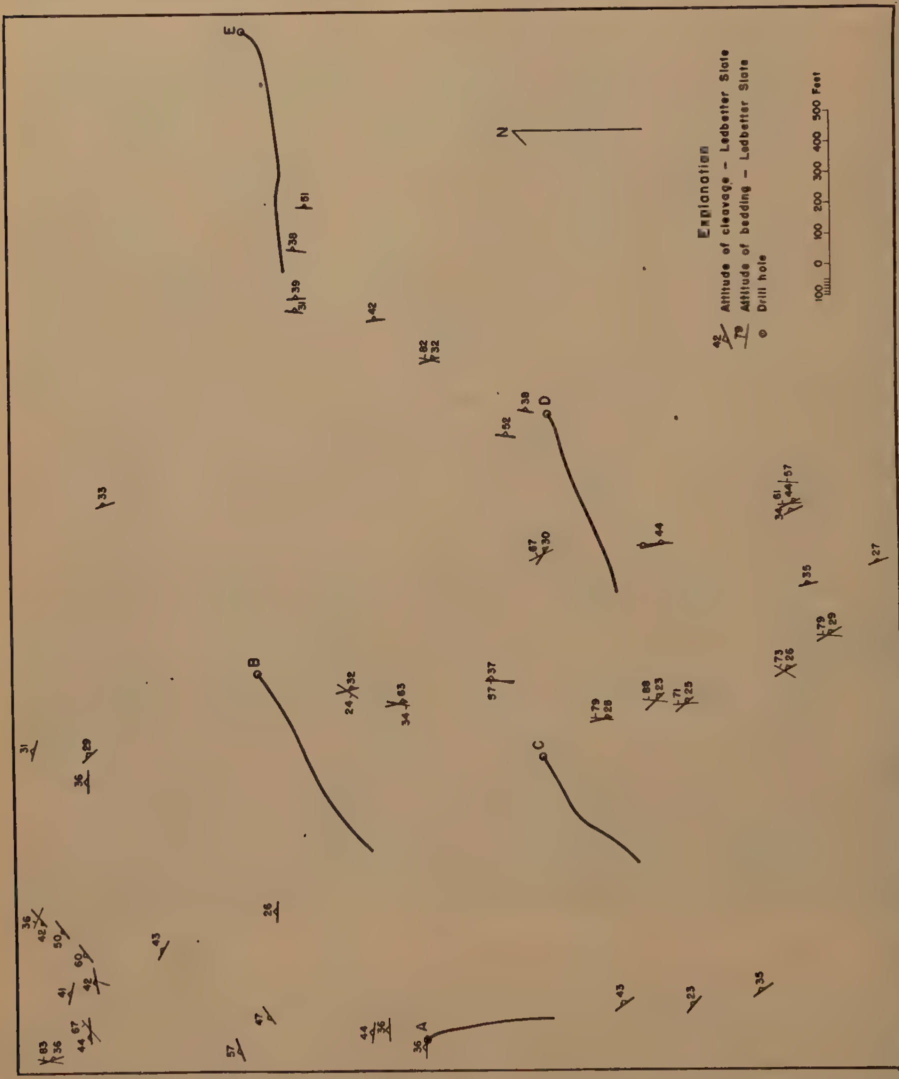
E
D
B
C
A
N
Explanation
Attitude of cleavage – Ledbetter Slate
Attitude of bedding – Ledbetter Slate
Drill hole
100 0 100 200 300 400 500 Feet

FIG. 3.—RELATIVE POSITIONS OF DRILL HOLES OF FIGS. 4, 5 AND 6, AND HORIZONTAL PROJECTIONS OF HOLES. Strikes and dips of surface exposures of bedding and cleavage are indicated.

## METHODS OF SURVEYING DIAMOND-DRILL HOLES

All holes drilled by the Bureau of Mines in the Metaline district since October 1943 have been surveyed, and the surveys have been jointly observed by the Geological Survey and the Bureau. The writers wish to acknowledge the helpful cooperation of Messrs. Evan Oscarson, District Engineer, Walter B. Cole, Project Engineer, and W. E. Walcott, Associate Engineer. Most of the details of procedure were carried out by Mr. Walcott.

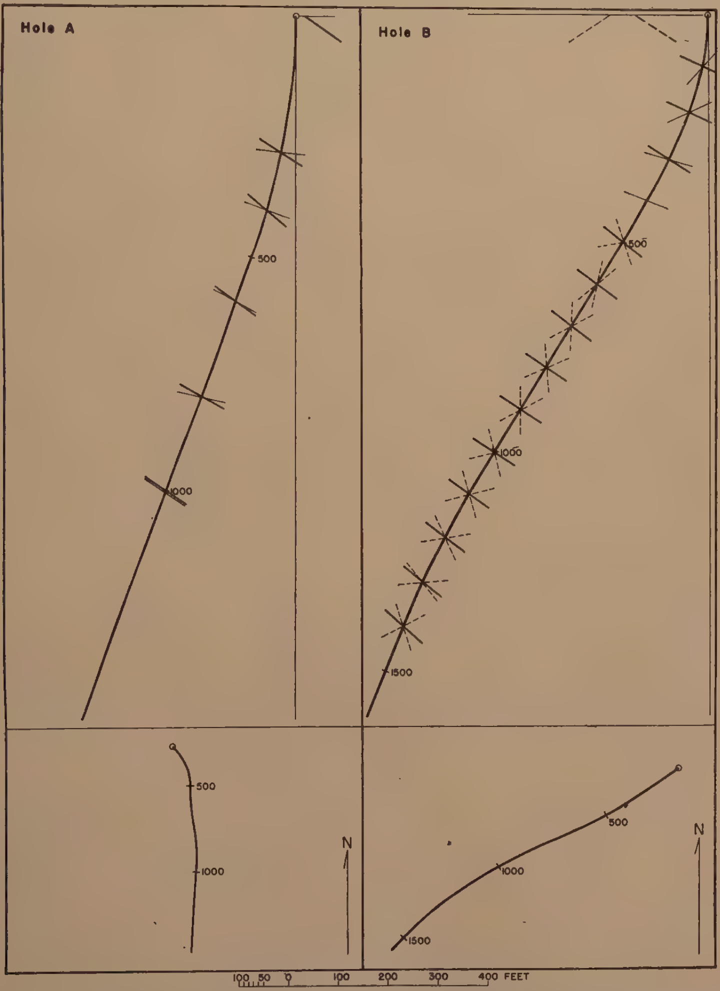

FIG. 4.—PROFILES AND PLAN OF DIAMOND-DRILL HOLES.

Heavy cross lines indicate dip of cleavage; light cross lines indicate dip of bedding. Dashed lines on surface indicate dips projected from surface exposures not directly on line of section; same lines on profiles indicate the two alternatives of position of bedding where data do not permit choice of which position to accept.

The amount of deflection was determined, for almost all observations, by the usual method of etching a glass tube with hydrofluoric acid. In the earlier surveys the Maas compass was used to determine azimuth, but the Carlson instrument has been used for later work, including all holes discussed in this paper. In a few instances the Carlson readings, unconfirmed by the etched tubes, were used to determine the amounts of deflection. Both the Maas and Carlson instruments require the use of gelatin solutions and a serious problem was encountered in getting the proper set of the gelatin under the temperature conditions prevailing. This aspect of the problem has been dealt with by the Bureau of Mines.

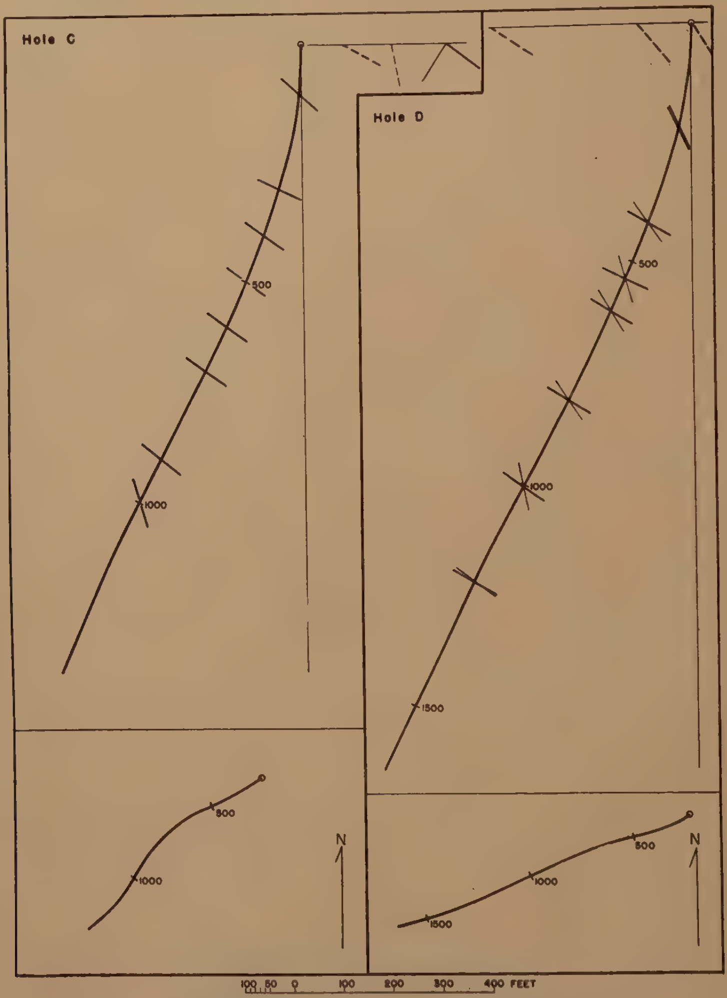

FIG. 5.—PROFILES AND PLAN OF DIAMOND-DRILL HOLES. Explanation of lines under Fig. 4.

For most of the work no goniometer was available for reading the acid-etched and Maas tubes, and methods were devised by the senior writer which gave results that the writers believe to be fully as accurate as those obtained by the goniometer. All readings of hole surveys recorded here were made by members of the Survey, and are in satisfactory agreement with readings made by the staff of the Bureau.

A chart for meniscus correction for the acid-etched tubes was made by the writers,* who wish to emphasize the importance of making up such a chart for any project, using the same materials as those actually used in the field, and employing the same method of reading the tubes. The maximum deflection correction was about 8½° for etches of about 45°.

Much plane-table survey work in the region has failed to disclose any local attraction and the writers believe that compass readings are dependable for azimuth determinations of drill holes.

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* See Appendix.

### AMOUNT OF DEFLECTION

All the holes were started vertically, but the surveys showed that while penetrating the slate they were deflected. In one instance a hole had deviated 35° from the vertical in 300 ft. and another was deflected 70° at a depth of 1000 ft. These were extreme cases; the usual maximum deflection was about 30° and such deviation was not indicated by any action of the rods or machines. Holes in limestone, whether altered rock or not, went remarkably straight. This has also been the experience of the mine operators in underground drilling. In deep drilling the surveys showed that the holes maintained their direction on passing from slate into limestone. The position of the slate-limestone boundary in the holes illustrated may not be indicated.

### CHARACTER OF THE LEDBETTER SLATE

Because of its sheared and distorted condition, the thickness of the Ledbetter slate is unknown, but it certainly reaches several thousand feet. The Ledbetter slate is a black, nearly uniform, fine-grained rock, which varies from a shale to a commoner, distinctly cleavable slate. Bedding is indicated in places by thin compositional bands but these are obscured in the slate. In the area covered by this report the slate contains a very few thin quartzite bands.

For purposes of studying textural relations and microscopic features related to cleavage and bedding, five oriented thin sections of slate from four specimens of core were studied. Specimens were selected that showed definite bedding laminations. Figs. 1 and 2 are micrographs taken of two of the thin sections. These sections show the rock to be composed of angular quartz, commonly from 0.01 to 0.04 mm. in diameter, embedded in an opaque ground mass and constituting from about 10 to 60 per cent of the rock. Calcite is present in all sections, and commonly amounts to about 15 per cent. However, one section of well laminated rock contains about 80 per cent calcite. Most of the remainder of the rock is black opaque material, which probably obscures many smaller grains of quartz and calcite. The highest power of the microscope fails to resolve these particles,

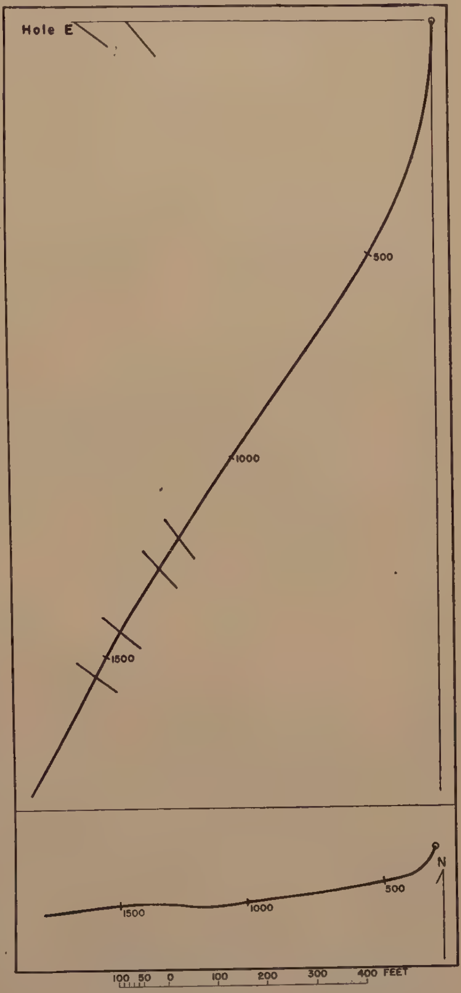


FIG. 6.—PROFILE AND PLAN OF DIAMOND-DRILL HOLES.

and masses as small as 0.0003 mm. are clotlike aggregates, entirely opaque and black. Sericitic material, in plates averaging about 0.006 mm. in width, is sparingly present. Most of the plates replace quartz and some others lie within the matrix, elongated parallel to the cleavage. The amount of sericite is quite inadequate to account for the cleavage and also to account for the normal alumina content of such a rock; therefore it seems that much of the opaque material must be aluminous and probably consists of clay minerals blackened by carbon. A small quantity of pyrite is present, mostly extremely fine-grained.

Quartz grains are sheared and plates 0.006 by 0.04 mm. are seen, which are separated by films of the black opaque material. It is concluded from the study of the thin sections that the cleavage directions are closely related to mechanical shearing of the rock and not to any orientation of secondary minerals.

## Illustrations of Diamond-drill Holes

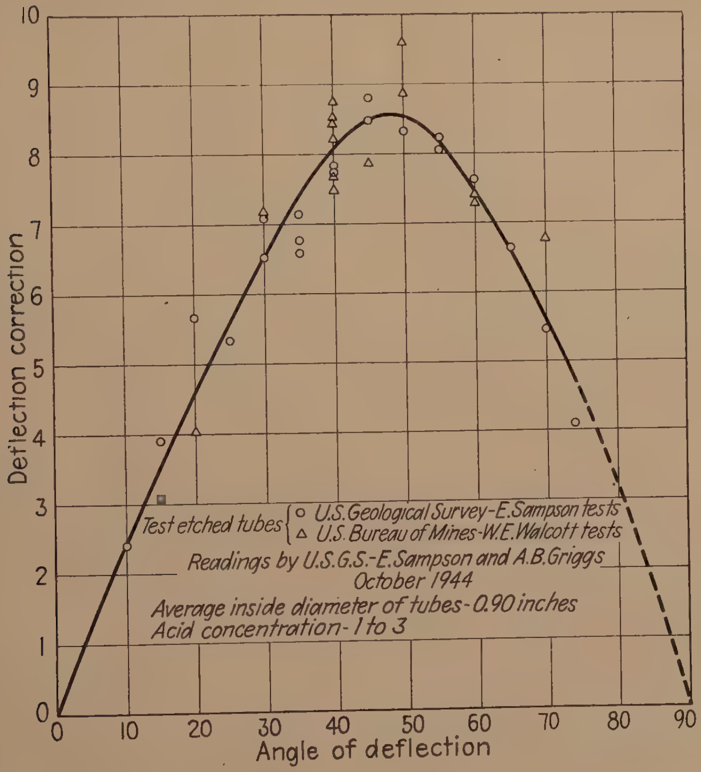
Fig. 3 shows the relative position of the drill holes illustrated in Figs. 4, 5, and 6 and the surface attitude of the cleavage and bedding of the slate. The trend in plan of these holes is definitely toward a normal to the strike of the cleavage of the slate. The attitude of the cleavage plotted along the profiles in Figs. 4, 5, and 6 further indicates that the holes tend to become perpendicular to this cleavage. It is to be borne in mind that the profiles are not true projections, but are drawn on a vertical surface passed through the plane of the hole. However, the profile of the holes illustrated is almost identical to a true projection.

In interpreting the attitude of the cleavage and bedding in the cores it is necessary to take careful account of the surface geology, because the true position of the core may be at any point of rotation about its axis. The core alone by visual inspection gives no criterion for determining the attitude of any plane surface unless the surface is perpendicular to the axis of the hole. The surface geology indicates that for all holes the strike of the cleavage is substantially normal to the direction of deviation, and the strike of the bedding on the whole is also normal. Therefore, with the strike essentially fixed, the core indicates two possible positions each of cleavage and bedding, and observed surface dips commonly afford the criterion for selection of one of these. The figures, except No. 4 (hole B), have been drawn with only the most probable of these indicated.

The determination of the bedding in the holes in question is not so satisfactory. For holes A and D, surface data indicate that bedding dips should be to the right, as shown in the figures of the profiles (Figs. 4 and 5) and this also accords with the alternative accepted for the cleavage. For hole E (Fig. 6) no satisfactory bedding laminae were observed. Surface observations near hole C (Fig. 5) indicate that the strike of the bedding is not perpendicular to the trend of the hole and that, therefore, the information on bedding is of little value, except that bedding planes are substantially inclined to the plane perpendicular to the hole and have no relation to the deflection. On the profile of hole B (Fig. 4), the two alternatives of bedding dip are shown. Strikes of bedding and cleavage are in moderate accord but, inasmuch as the cleavage is nearly normal to the axis of the hole, no criterion is available to select the true direction of dip of the bedding. It is clear, however, that bedding has not controlled deflection, whichever alternative of dip of bedding is selected.

The data indicate clearly that holes have shown a very marked tendency to deflect so as to become normal to the cleavage of the slate.

APPENDIX.—*Chart for Meniscus Correction for Etched Tubes*



# A Résumé of Bureau of Mines Experience with Oversize Core Barrels

By J. R. Thoenen,* Member AIME

(New York Meeting, March 1947)

## Abstract

The Bureau of Mines has used various sizes of core barrels above 2 in. and below 10 in. in diameter to core manganese, potash, coal, brown iron ore and bauxite. The paper describes in some detail the methods and types of equipment used to obtain maximum sampling accuracy.

## Introduction

On December 11, 1946, R. D. Longyear, in his capacity as Chairman of the Subcommittee on Exploration Drilling, wrote the author requesting him to prepare a paper presenting the collective ideas of engineers in the Bureau of Mines on the design, standardization and application of core barrels using diamonds or other hard cutting media for drilling holes larger than two inches and less than ten inches in diameter.

In subsequent correspondence it was indicated that similar papers covering the experience of other government agencies would be solicited and all such papers received would be combined in a symposium on the subject for discussion at the March 1947 meeting of the Mining Methods Committee AIME.

In soliciting such information from other Bureau engineers, it developed that most Bureau experience had been confined to standard core size, two inches or less in diameter, and that larger sizes had been used only for special purposes. It is hoped the data contained herein will be of help to other operators in solving problems encountered in the field.

## Acknowledgments

This paper was made possible by the request of Lowell B. Moon, Chief, Mining Division, Bureau of Mines, to all Division engineers to send to the author any data they had on the subject.

Engineers of the Mining Division contributing were:

M. H. Kline, Chief, Raleigh Branch.

Paul T. Allsman, Chief, Salt Lake City Branch.

J. H. Hedges, Chief, Tucson Branch.

C. H. Johnson, Chief, Rolla Branch.

A. M. Evans, Mining Engineer, College Park Branch.

Wing Agnew, Engineer in charge, Mount Weather, Virginia.

Andrew Brown, Mining Engineer, Tuscaloosa Branch.

In addition Albert L. Toenges, Supervising Engineer, Bituminous Coal Mining Section, Pittsburgh, contributed his experience in coring coal beds.

The author's experience in the use of cores within the limits stated has been confined to the brown iron-ore work described by Brown and coring bauxite and clay in Arkansas.

## Design and Standardization of Core Barrels

Bureau engineers are primarily interested in obtaining core that will best represent

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Published by permission of the Director, Bureau of Mines, United States Department of the Interior. Manuscript received at the office of the Institute Oct. 15, 1947. Issued as TP 2385 in Mining Technology, May 1948.

* Chief, Tuscaloosa Branch, Mining Division, Bureau of Mines, University, Ala.

the ground drilled. The design of the core barrel used to get that core and whether or not it is a standard size is of secondary interest. On the other hand, Bureau engineers encounter many unique problems in the field and have been instrumental in designing special equipment to fit particular needs. These, however, will seldom fit in any plan of standardization. In the author's opinion, any standard core size selected by the Core Drill Manufacturers Association will be satisfactory for routine drilling in the province of Bureau engineers.

## APPLICATION OF CORE BARRELS

It is probably in the field of application of core barrels to mineral development that the experiences of Bureau engineers will be of most assistance. This also may involve change from standard design or use to fit a particular problem but that same change in design or use may be applicable to other fields than the one for which it was initiated.

## EXPERIENCE OF BUREAU ENGINEERS

### *Manganese*

In drilling for manganese in Page County, Va., it was necessary to drill through overburden consisting of soil and sandy clay and recover a representative core of the nodular manganese imbedded in a clay stratum. To accomplish this, a 10-in. churn-drill hole was drilled to the ore bed and cased with 8-in. pipe. The ore bed was then cored with a diamond drill. In commenting on this work, Agnew states:

> We used a swivel-type, double-tube, 10-ft core barrel cutting a 5.5-in. core. It had a basket-type core puller riveted to the bottom of the inner tube . . . On top of the core barrel was a calyx approximately 2 ft deep which caught a considerable amount of sand and pebbles. The outside diameter of the barrel was 6.5 in. and the bits were usually stellited saw tooth but occasionally hand-set diamond bits were used.

Commenting on the same operation, Kline says:

> Clay fairly free of sand cored very well and was drilled in 5-ft runs. In decomposed chert, the core recovery was poor because the chert ground to sand and gravel in the barrel. Hard manganese nodules slowed the drilling and caused a scouring action on the core allowing some of the clay to be washed away.

For similar work in the future Agnew recommends that the core size be reduced so that the diamond-drill core barrel can be used through a 6-in. casing.

### *Potash*

Potash salts are soluble in water and because of this a heavy brine was used as a drilling medium in the Bureau's investigation of potash resources in the southwest. Even with the use of brine, it was feared some core would be dissolved and lost. To obviate this as far as possible and obtain satisfactory samples, it was decided to increase the core size to 2.5 in. in diameter with the anticipation that the recovered core after partial solution would still be large enough for sample purposes.

In presenting this experience, Hedges refers to a paper by Jas. S. Roth,[1] to which the reader is referred for details, but the following abstracts will be of pertinent interest. The core barrel was long enough to permit taking 16 ft of core in a single run. Core diameter was 2.5 in. The barrel was modified from standard design and is described by Roth as follows:

> The head of the barrel is of the standard type, with the inner barrel mounted on a ball-bearing swivel . . . The solution is fed to the space between the two barrels . . . In the modified barrel, holes are bored in the bit and the drilling solution is made to feed through these holes to the cutting face of the bit without coming in contact, at any point, with the core. The clearance between the bit and the inner barrel is reduced to a point only

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[1] References are at the end of the paper.

sufficient to allow free rotation . . . less than 1⁄32 in. and in some barrels the inner tube is recessed into a ring cut into the bit.

Fig 1 is an idealized sectional view of this core barrel.

Regarding the cutting medium Roth states:

> Several holes were drilled with sulamite, a tungsten alloy, set in the bits instead of diamonds. The sulamite . . . had a tendency

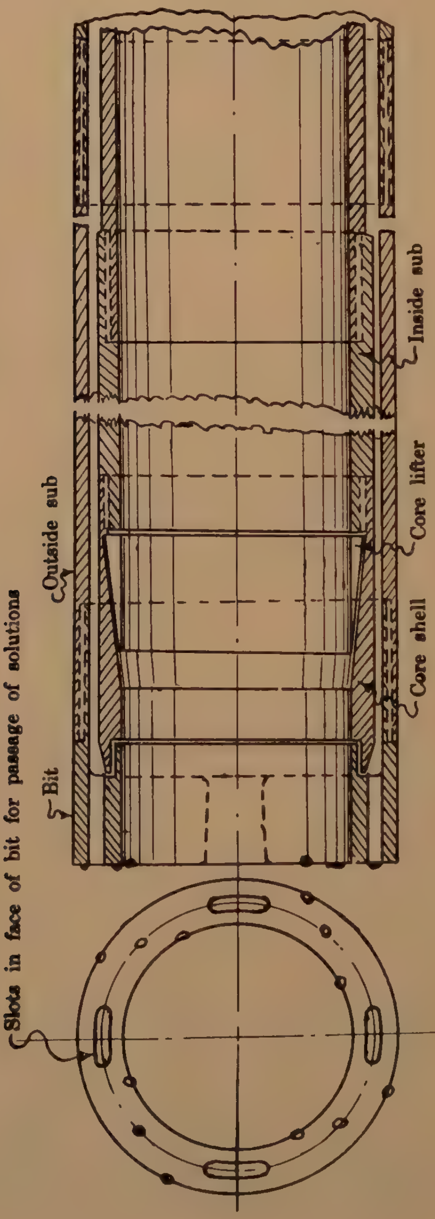

FIG 1—SKETCH OF BIT END OF DOUBLE-TUBE CORE BARREL FOR SALT CORING.
Note: this is a diagram to illustrate the principle and is not a drawing of an actual bit.

to tear the salt crystals . . . and . . . after a comparison . . . with diamonds . . . the use of diamonds was specified.

Allsman referred to an article by Severy[2] which describes the use of a wire line core barrel in connection with standard oil-well drilling equipment, and since oil-well equipment was ruled out for this symposium the reader is referred to the published article for details. However, it is only fair to Allsman to quote his letter to the author as follows:

> I am of the opinion that at several places where we have used *NX* and even *BX* diamond core barrels we could have done a cheaper job with a wire-line core barrel adapted to drill holes up to 2000 ft from surface. I feel certain that in drilling holes involving taking cores larger than *NX* the principle of the wire-line core barrel could be adapted and give cheaper and more rapid drilling.

### *Coal*

In the Bureau's investigation of deep-lying coal deposits in Utah during the war, the standard diamond-drill core sizes were too small for obtaining samples of coal large enough for satisfactory laboratory tests. To obtain a sample of sufficient size would have required drilling many long holes involving thousands of feet of drilling in waste overburden. It was therefore decided to contract for one or more holes started large enough to permit taking an 8-in. diameter core in the coal bed. Relative to the experience gained by this drilling, Toenges is quoted as follows:

> The Pennsylvania Drilling Company designed a special, 10-ft, double core barrel with diamond bit to do this work. The bit and barrel were similar to standard design . . . However, there were a number of 'bugs' which developed . . . For instance, the heavy journal and bearing on which the inner core barrel hung caused some trouble. This was a ball bearing (7/8-in. balls) and the inner barrel dropped out at a depth of 900 ft in one hole due to weakness of the journal. The inner barrel and balls were recovered . . . and the weakness overcome by redesign . . . Four large diamonds were set on the outside of the outer core barrel about 6 in. above the bit to act as reamers. Twenty odd diamonds were set in the bit. With the improved barrel, five holes were drilled taking an 8¼-in. core. We usually drill down to within 100 ft of the coal with a churn drill or rotary bit. In every hole drilled using this equipment, core recovery was 100 pct.

The core barrel used in this work is illustrated in Fig 2.

### *Soil Cores*

Evans describes the method of taking soil core samples by the United States Engineers as follows:

> The core barrel (for recovering samples in dam foundation testing) was 9 in. id with an effective length of 18 in. Onto the top was fitted a plug with connection to the drill rod. Into the core barrel was fitted an inner barrel made of galvanized sheet iron which was 18 in. long and fitted almost flush with the walls of the outer barrel . . . Just below this inner barrel was fitted a spring core lifter of the same diameter as the drill bit to which a number of thin strips of steel radiating toward the center were attached . . . Below the core lifter was the saw tooth bit with from 4 to 6 long, thin, stellite set teeth.
>
> These tools were lowered into the hole and the hole deepened 18 in. The tools were pulled and the bit and core lifter removed. A wooden block 1 in. thick was inserted in the bottom of the inner barrel. The top of the outer barrel was then removed and a similar block fastened in the upper end of the inner barrel.
>
> Gelox and aquagel were used in the circulating medium.

### *Fluorspar*

Johnson commented that experience in drilling fluorspar in Kentucky and Illinois indicated that increasing bit size from *BX* to *NX* did not yield higher core recovery in that field and his field men felt that: "The general belief in improved recovery via larger core should be expressed only with some qualification."

### *Alabama Brown Iron Ore*

The customary practice in drilling Alabama brown iron ores involves the use of churn drills. In the past this type of drilling has been confined to placing a few holes ahead of the power shovels to guide their advance. Recently some attention has been given to the use of the churn drill to block out ore bodies ahead of mining. In the belief that the truck-mounted rotary core drill might offer some advantages over the churn drill in this field, the Bureau laid out a program for this type of investigation in the Woodstock, Russellville and Shelby districts in Alabama. Since drilling conditions differ in the three districts and the methods developed to obtain maximum core recovery varied with those conditions, each district will be discussed separately for clarity.

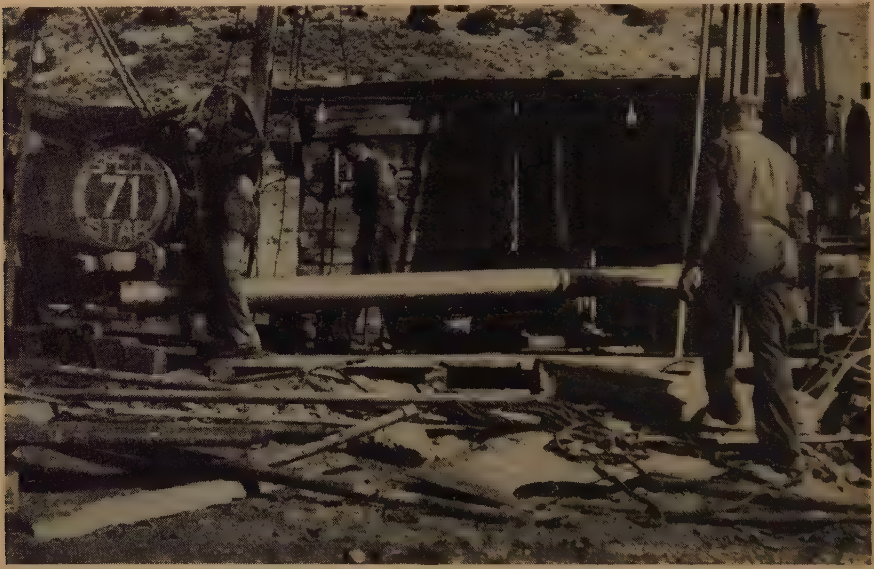

FIG 2—AN 8-INCH DOUBLE-TUBE CORE BARREL USED TO CORE DEEP UTAH COAL.

*Woodstock District*—The ore in the Woodstock district is largely in the form of nodules averaging not over 1½ in. in diameter, although some larger "dornicks" were encountered. The ore lies in and on a replacement clay covered by irregularly distributed sand and clay beds with some thin ledges of ferruginous sandstone. The holes ranged from 15 to 280 ft in depth with cores taken as shallow as 3 ft and as deep as 120 ft.

Brown describes the drilling as follows:

> The first efforts at coring were made with a standard double-tube *N*-core barrel and were uniformly unsuccessful. A 3-in. core barrel similar to those used in bauxite drilling was then tried. Recovery improved immediately, but two difficulties were met. First, the sandy parts of the core washed away; second, the clay parts had a tendency to stick in the inner tube, with the result that only about 18 in. of core were recovered on any run, no matter what its length. Tests of the clay showed that it was not bentonitic, and that therefore the trouble was due not to swelling, but to packing by the water in the tube.
>
> Washing of the sand was at least partly overcome by (1) dropping the bottom of the inner

tube to within ⅛-in. of the top of the teeth of the bits, and (2) installation of a separate transmission for the drill pump. The drill, as first used, had no such transmission, and the water pressure went up and down with the drill speed. Core recovery on this machine increased 10 to 15 pct after the change was made.

Packing of the clay core was overcome by building up the bits to cut from 2¾ to 2⅞ in. inside clearance, as compared with the 3¼ in. id of the inner tube. When runs of 3 to 5 ft were used, this took care of the packing problem.

The core barrel which proved most satisfactory at Woodstock carried an inner tube 5 ft long and 3¼ in. id. The outside diameter of the outer tube was 4½ in. exclusive of reamers down the sides; the bits cut 2¾ in. inside and 4⅞ in. outside clearance. The most satisfactory bit carried 8 built-up teeth, about ¾-in. high, faced with borium, and set at an angle of 60° with the radius, the inner edge of the teeth leading in order to throw the cuttings toward the wall of the hole. No core catchers were used.

Coring procedure depended on the material being drilled. In sand, best results were obtained by running the drill fast with a minimum of water; in clay, low speed and high water pressure were necessary; in the ore zone, where most of the matrix is clay but some sand is present, speed and water pressure were, as a rule, both kept low. Weight feeding was the almost invariable rule. But it should be pointed out that different drillers obtained satisfactory results by somewhat different methods. One rule, however, was seldom broken; that of drilling about 2½ ft, cutting off the water, and blocking for about 6 in.—this making up the 3-ft run which was insisted on unless the 3-ft point came at a place where blocking was impossible. In this case, the 5-ft barrel provided a margin of safety.

After the experimental stage was passed, core recoveries of from 70 to 80 pct were obtained with much of the loss being accounted for by sand beds. Few cores were dropped during the drilling, as pulling was not particularly difficult. Excellent results were obtained in sand standing above the water table, in the clays, and in the ore zones. The sand in this area is very fine and below the water table is much like quicksand; this material was never cored satisfactorily.

The lump ore, in its clay or sand-clay matrix, gave little trouble in coring though it is recognized that the drill does not actually cut the ore, but rather pushed the lumps either inside or outside the barrel and the sampling may be inaccurate to this extent. The dornicks were cut satisfactorily, if very slowly, with the borium-faced bits, using light pressure and much water. The structure of brown-ore dornicks is such that they are chipped off rather than cut smoothly by this method, and the cores present a jagged appearance.

*Russellville District*—The ore in the Russellville district was similar to that at Woodstock except that in all but one deposit drilled the nodules were larger and predominantly "lump" size but not as large as "dornicks." The size probably averaged from 1½ to 6 in. in diameter. The one deposit excepted contained only fine ore less than one inch in diameter.

The material encountered from the surface down was (1) coarse red sand with predominate chert gravel to 3 in. in diameter (Fig 3); (2) a bed of conglomerate formed of chert gravel cemented with brown iron ore or ferruginous sandstone from 1 to 3 ft thick, (3) the ore zone containing ore in a matrix of clay with some sand beds, and (4) a blanket of replacement clay.

Brown describes the drilling as follows:

> The obvious method of drilling this material is to fishtail to the conglomerate bed, case, and core the conglomerate and the underlying ore. This was tried, the casing being washed down in the first attempts. This process washed so much gravel from the sides and into the bottom of the hole that it was impossible to get the casing to the bottom. Casing was then driven and the results were similar in that gravel falling in prevented the casing reaching bottom. Both 5- and 6-in. casings were used in the trials. Both 3-way bits and rock bits were tried inside the casing with the idea of breaking up the gravel so it could be washed out, but the material was not solid enough for this treatment and the attempt failed. At-

tempts to clean the hole with bailers were also unsuccessful, though it is possible that a bailer could be devised which would be successful in handling the large material.

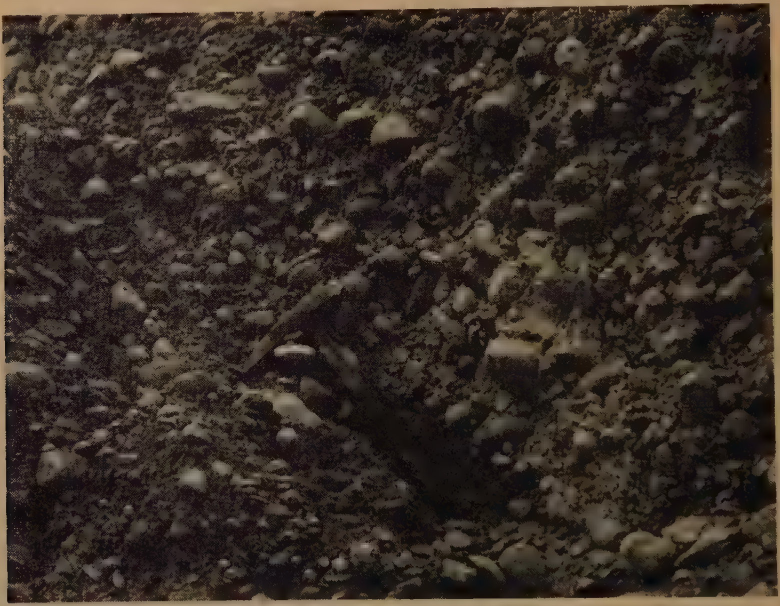

FIG 3—SAND AND CHERT GRAVEL SURFACE IN THE RUSSELLVILLE DISTRICT.

The method finally adopted possessed certain undesirable features, but it did make possible the obtaining of satisfactory cores. The holes were drilled through the sand-and-gravel overburden with a mixture of aquagel and lime, made as thick as the pump would handle. This thick mud cemented the sides of the hole to a degree greater than had been expected, and in addition floated all but the large gravel when the hole was washed out under high water pressure. In actual coring, thick mud was used also and the drilling was done with the mechanical pressure feed, as fast as possible and with high water pressure. The 2½-ft cut and 6-in. block were used, and the rods were pulled immediately upon completion of the hole to the depth desired. Pulling was invariably a slow and difficult job, because the hole was never cemented solidly enough to prevent gravel from falling from the sides to the top of the core barrel. Some cores were dropped during the pulling process, but the core recovery during the latter part of the drilling was more than 70 pct.

*Shelby District*—Drilling conditions at Shelby were much simpler and core recovery averaged 95 pct. From the surface down the materials drilled were (1) a few feet of top soil, and (2) stiff clay containing the ore and resting on limestone, dolomite or slate.

No aquagel or casing was necessary and the clay was cut slowly with medium water pressure without trouble.

*Other Types of Equipment Used*—In early attempts to determine the best equipment and method to use in coring at both Woodstock and Russellville various designs of both bit and barrel were tried. While they were not as successful as the methods adopted they will be described briefly since negative results are frequently as informative as positive.

One core barrel 8 ft long taking a 3-in. core and fitted with a long, standard tapered-type core catcher was tried. The bottom of the inner tube was 2 in. above the teeth, which were short and set radially on the end of the bit. This did not prove satisfactory and the design was altered by dropping the inner barrel to the top of the teeth and replacing the tapered core catcher with one which combined both the basket and spring types. This core catcher seemed to hinder the core from entering rather than holding it in the barrel. This barrel was not popular with the drillers because of its length and weight and was superseded by the 5-ft barrel without core catchers already described.

A 10-ft barrel of standard design but cutting a 6-in. core was also tried but was discarded because of its weight and because of the difficulty in pulling from the uncased holes.

A 10-ft barrel taking a 5-in. core and fitted with a spring core catcher was tried successfully at Woodstock but it failed dismally at Russellville because of the difficulty in pulling.

*Mountain Creek District*—The Mountain Creek district was a fourth area drilled. While no commercial ore was developed it is of interest because of a still different set of drilling conditions. From the surface down, the materials encountered were (1) the soil zone averaging about 6 ft thick containing a great deal of loose gravel from 1 to 3 in. in diameter and many large dornicks of iron ore, and (2) the bed rock of weathered mica schist cut by numerous pegmatite dikes.

Regarding this drilling, Brown says:

> Although considerable difficulty was anticipated in coring the weathered schist, it has turned out to be the easiest coring we have done. The 3-in. barrel which has been used in all other localities was used, with 3-ft runs. Coring was done with the mechanical pressure feed rather than by tool weight only, at low speed and low water pressure. Under this procedure, the nonmineralized portions come out as a rather tightly-packed core of sand, indicating that the drill water disintegrates the rock to a large extent. The mineralized parts, on the other hand, come out as solid cores with very porous structure.

### *Arkansas Bauxite*

Prior to the Bureau's program for drilling the Arkansas bauxite field drilling contractors had tried various methods and several bit and core-barrel designs. The method finally adopted by them was to fishtail to the bauxite horizon, case the hole, and core with standard *NX* tools, using fuel oil instead of water as a drilling medium.

The reason for using oil instead of water requires some explanation. Bauxite is easily dissolved or disintegrated by water. It has the further peculiar characteristic of adhering tightly to a clean steel surface. Consequently with the core barrels used, in which the lower end of the inner barrel stopped 2 or 3 in. above the bit teeth, the initial core entered the inner barrel and adhered to the tube. Drilling water would then disintegrate additional core, which could not enter the lower end of the inner barrel because of the bauxite plug. When fuel oil was used instead of water, this trouble was overcome and coring was eased materially.

Bureau specifications required recovery of 80 pct of the interval ordered cored under penalty of loss of payment for the entire hole.

While the use of oil increased core recovery it created a fire hazard and there were still too many holes lost because of insufficient core recovery.

About this time the Metallurgical Branch of the Bureau asked for larger samples for ore-dressing test purposes than could be obtained with the *NX* cores. Accordingly, a 6-in. barrel of standard design was obtained. This barrel was successful without the use of core catchers and it was found that core recovery with water was equal to that with oil. In using this barrel the core was blocked at the bottom by running the drill without water for a few minutes.

The success in the use of this large barrel led to experiment with other sizes until the standard 10-ft, double-tube barrel without core catchers and taking a 3-in. core was adopted. Experiments showed that with this size core water could be used as a drilling medium and the oil was discarded.

The inner barrel as finally adopted was carried on a swivel at the top and extended to within ½ in. of the bottom of the saw teeth in the bit. The teeth were set radially on the outer barrel and the inner gauge was ⅛ to ¼ in. less than the diameter of the inner barrel. This cut a core substantially smaller than the inner barrel and prevented packing. The outer gauge of the teeth was from ¼ to ½ in. greater than the diameter of the outer barrel to provide adequate hole clearance. The outer barrel was fitted with four stellite ribs both behind the bit and at the top of the barrel. These served to align the barrel in the hole and to reduce barrel wear due to abrasion.

The teeth on the bits were of hard metal "stellite" or "borium" stuck on the end of the outer barrel. No attempt was made to trim the teeth except to grind them to the proper inner and outer gauge.

Using this type of bit and barrel, this project completed roughly one and a half million feet of completed hole in the drilling of which the contractor was penalized for about 10 pct of the actual footage drilled because of less than 80 pct core recovery.

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1. Jas. S. Roth. Bur. of Mines *I.C.* 6156 (1929).
2. C. Luther Severy: Salts Cored Successfully with Oil Field Equipment. *Eng. and Min. Jnl.* (Aug. 1946) **147** (8).

# Recent Modifications of the Hall-Row Wedging Technique in Diamond Drilling

By Burton H. Boyum,* Junior Member AIME

(New York Meeting February 1948)

## Abstract

Greater depth of exploration drill holes and increasing cost of drilling have been principal factors in reviving interest on the Marquette Iron Range in controlled directional drill-hole deflection. This deflection is obtained by the application of the Hall-Row wedging technique. Modifications were made in the technique and equipment in the effort to simplify and strengthen the procedure. The practice is similar in many respects to the oil well type of deflection with a whipstock.

## Introduction

The deflection of diamond drill holes by wedging was a widespread practice in Canada, South Africa and the Lake Superior District as early as 1907 to 1920. Several basic schemes were used, with local variations.[1] The deflecting wedge assembly was the idea of Mr. Welldon and Mr. Payne-Gallwey operating under the supervision of Mr. William Gallagher of Johannesburg, South Africa. All schemes employed some means of plugging the old hole. A footpiece of various styles was placed in the hole on the plug. In some methods, the orientation of this piece was determined before placing the deflecting wedge. Some techniques permitted continuing the hole the same size after wedging, while others necessitated a size reduction. Interest in wedging diamond drill holes in the Lake Superior District waned after 1920. The Hall-Row patented system for deflecting holes was developed in Canada about 1918.[2]

Interest centers on the wedging process for the following applications:

1. Maintaining a given course and inclination of the drill hole.
2. By passing an obstruction in a drill hole.
3. Resampling an ore body.
4. Exploring an area overlain by surface conditions not favorable to drilling.

During the period of 1913 to 1920, The Cleveland-Cliffs Iron Co. used the wedging technique under a variety of conditions. In general, the practice was not applicable again until 1946. Meanwhile the Hall-Row system of drill-hole deflection survived years of application as the best technique available.

In the intervening years, our depth of exploration has increased as well as the overall cost of drilling. As exploration becomes deeper, there is the possible economy of cutting the ore body at a moderate distance from the parent hole without redrilling from surface. Thus a branch hole started near the mid-point of a deep hole can reach the ore horizon at a considerable offset from the parent hole. This permits a greater interval between surface drill holes as drilling becomes deeper. Our interest revived in the subject and a wedging program was initiated.

## General Description

### *Results*

The maximum theoretical angular deflection possible, based on the deflecting

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Manuscript received at the office of the Institute Dec. 1, 1947. Issued as TP 2410 in Mining Technology, July 1948.

* Assistant Chief Geologist, The Cleveland-Cliffs Iron Co., Ishpeming, Mich.

[1] References are at the end of the paper.

wedge used in a *BX* hole is 1°50′. Our overall average deflection from a total of 21 wedges set to date, has been 1°6′. This has included a wide variety of conditions of cutting the stratification. The minimum deflection obtained was 0° and the maximum was 2°, based on the precision of recording. Some holes have deflected progressively by the use of only one wedge. Usually, in such cases, the objective has been reached without using additional wedges.

### *Hole Plugging*

Generally, our drill holes are plugged from the bottom up to the point of deflecting the branch hole. This is done with common cement. Plugging may be done with a shorter cement plug or by filling the holes with old drill rods or wooden poles. We prefer cement as it precludes the possibility of ground water flowing into later mine workings through the open drill hole. Also, old drill rods may deflect any magnetic surveying device used in the branch hole.

A short wooden plug made of pine or fir is placed on the cement plug. Our first plugs were 2 ft long but were changed soon thereafter to a 3-ft length. They are turned on a lathe to the size of the bit-gauge ring. Longitudinal fluting is cut to permit passage of the plug through the standing water while being lowered into position. A diagonal saw-cut is made at mid-section of the plug and the pieces nailed together with small nails (Fig 1).

The wooden plug is lowered with an open rod down to the cement or other plug. It is sheared by the drill-machine hydraulic swivelhead. It could be sheared also by striking the rods on an old coupling with a sledge hammer. The rods are then pulled. The plug swells in the standing water and fills the hole. It is best to proceed at once with the drive wedge, especially if there is any mud or caving material in the hole.

### *Drive Wedge and Clinometer*

The next step is to lower the drive wedge onto and into the wooden plug. This wedge is attached to the lower half of the clinometer (Fig 2 and 3) with two brass or bronze pins. Note chisel-like edge on the bottom of the wedge so that it will sink into wooden plug. A glass tube or vial is partly filled with hydrofluoric acid, wrapped in paper and inserted in the clinometer. The coupling is wicked and tightened. This assembly is connected to the drill rods and lowered to the plug. The pins are sheared, as with the wooden plug, and the test allowed to set for an appropriate interval of time (½ to 1 hr). The clinometer is then pulled, allowing the drive wedge to remain in the wood plug.

### *Drill-hole Orientation*

The Hall-Row system is noted for its simple direct scheme of operation. The deflecting wedge can be oriented with reasonable assurance of its direction at least to the same degree of accuracy as the directional survey of the parent hole. If use were to be made of the wedge to bypass an obstruction or resample an ore body without regard to direction, the clinometer may be omitted. However, most applications of the Hall-Row technique require directional deflection.

It is assumed that the parent hole has been surveyed near the anticipated position of the wedge. Generally this has been done as routine practice. The Cleveland-Cliffs Iron Co. surveys each surface hole at 200-ft intervals with the Maas compass and a glass tube with hydrofluoric acid. Thus the deviation of the hole is known at the position of the plug. The test in the clinometer lowered with the drive wedge shows the position of the "step" of the wedge. The marks, 180° apart, are made on the top of the glass tube. One mark is an

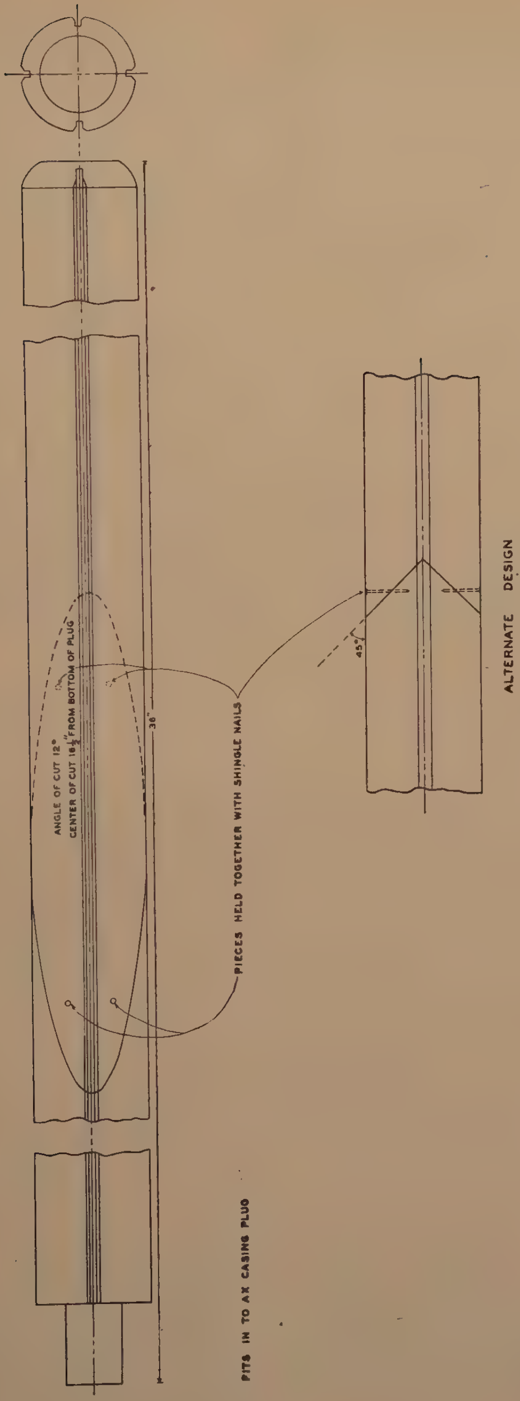

ANGLE OF CUT 12°
CENTER OF CUT 16½" FROM BOTTOM OF PLUG
36"
PIECES HELD TOGETHER WITH SHINGLE NAILS
FITS IN TO AX CASING PLUG
45°
ALTERNATE DESIGN

FIG 1—WOOD PLUG.

*X* mark (Fig 2). The tube is placed carefully in the clinometer so that the *X* mark coincides with a punch mark on the clinometer. This punch mark lies on the back center line of the "step." The course of the plane of inclination of the hole is known from the previous survey. From this, the course of the two marks on the glass tube is established.

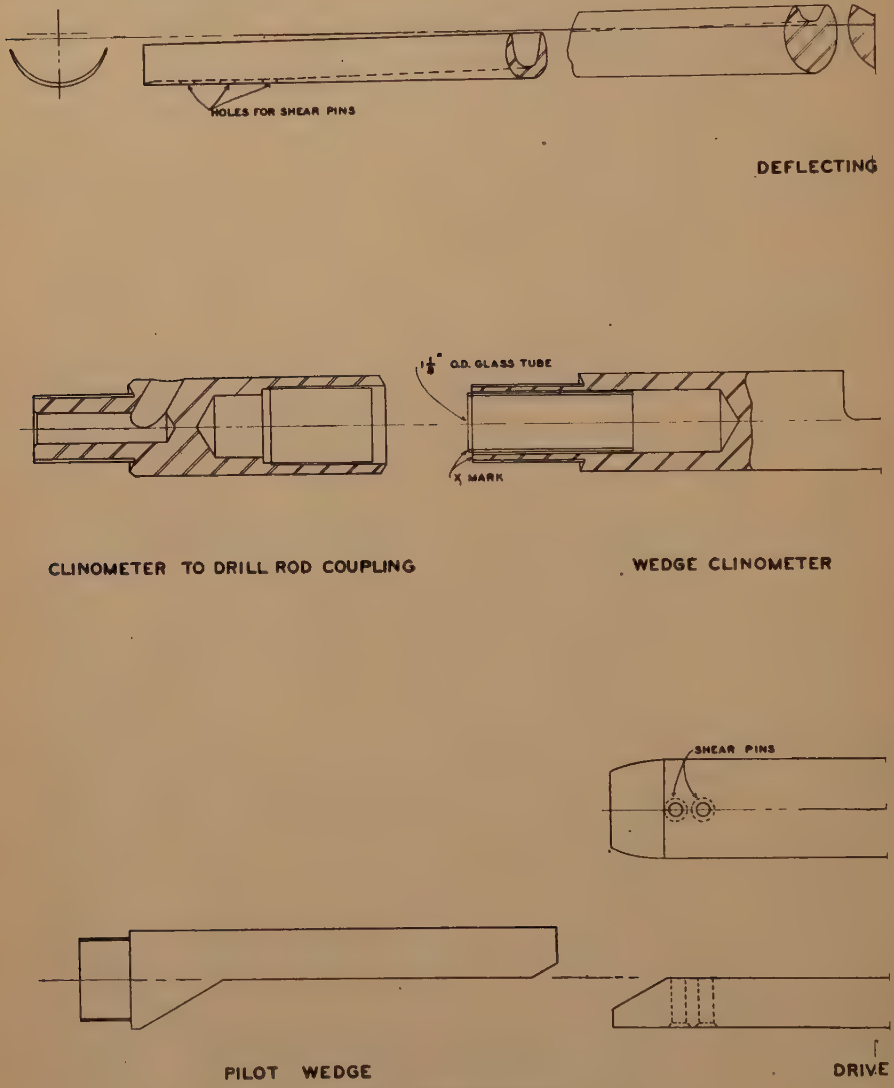
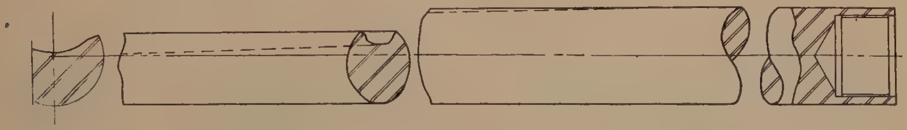


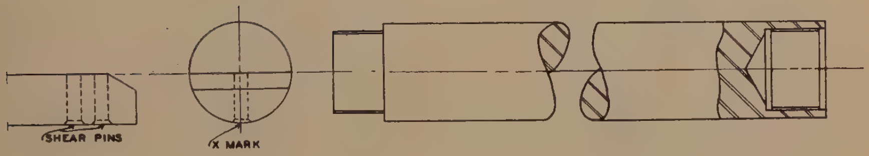
Fig 2—Wedge parts.

### *Pilot Wedge and Deflecting Wedge*

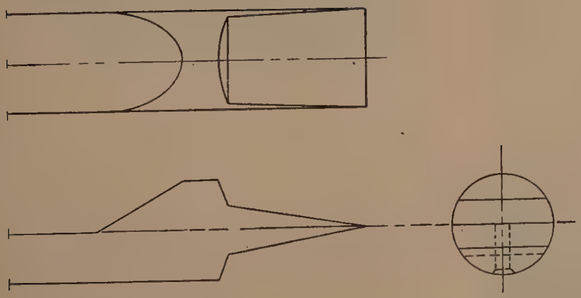
The pilot and deflecting wedges (Fig 2) are screwed together. Our practice is to set the deflecting wedge in a vise. The pilot wedge is threaded into the deflecting wedge up to the last turn where the flat of the step is paralleled to a surface laid on the wings of the deflecting wedge. A chisel



WEDGE



WEDGE EXTENSION



WEDGE

Fig 2—*Continued*

mark or two punch marks then is made. A full circle protractor (Fig 4) is placed on the full portion of the deflecting wedge. The mark on the pilot wedge is equivalent to the mark on the clinometer and the *X* mark on the glass tube. The azimuth of the line of these marks is known from the survey in the parent hole. Hence, it is a simple matter to orient the pilot wedge on the protractor. A second chisel mark is made. A quarter-inch hole is drilled through the two wedges. A piece of 4-in. spike is driven through the hole and sawed to length. The ends are peened smooth and checked with a bit ring. Caution must be observed to make certain that the pin does not lie in the axis of the wedge.

Our practice is to have two of the technicians read the tests and compute independently the angular setting. This check system is followed all the way through the operation including measurements of the wedge assembly, to know whether the wedge is seated properly. Subsequent drill-hole surveys of the branch hole generally confirm the accuracy of the technique.

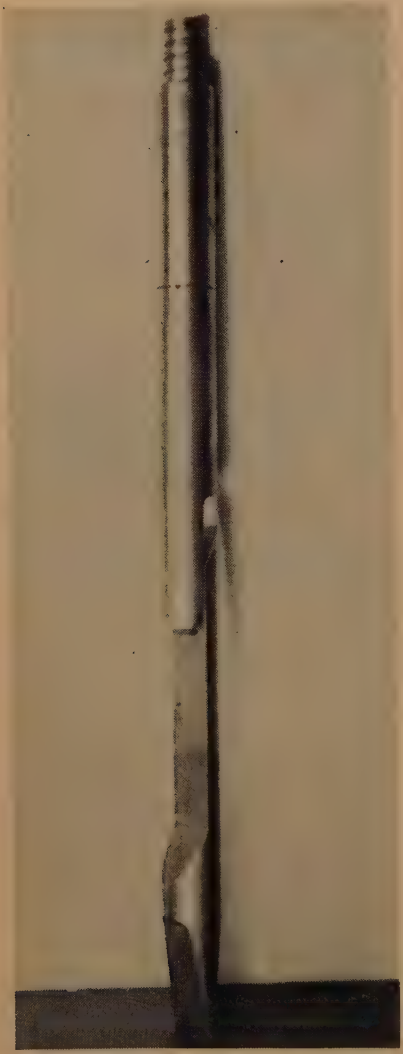


Fig 3—Wedge clinometer assembly.

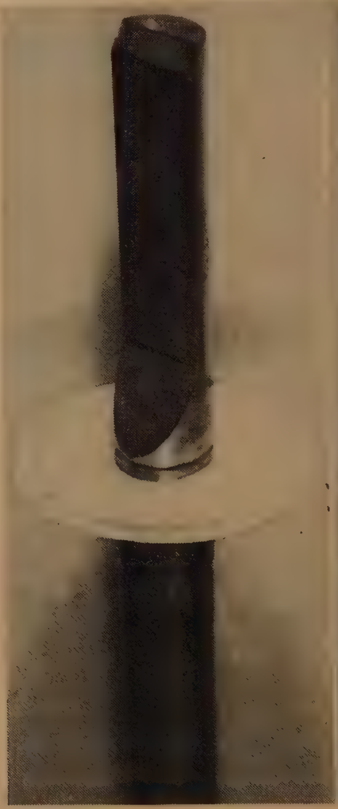


Fig 4

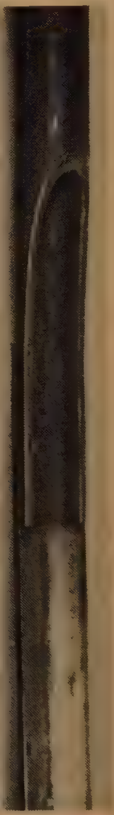


Fig 5

Fig 4—Full circle protractor.
Fig 5—Half drill rod pinned to deflecting wedge.

As expressed by Mr. Charles Trotter, the writing of the instructions concerning the orientation of the clinometer and pilot wedge, is a difficult task if the description is to be brief and lucid. The writer regrets that the above description is wordy and asks the indulgence of the reader in studying the description in conjunction with Fig 2. If the Hall-Row wedge is to be used to straighten the hole, or to maintain it in a vertical direction, the above procedure is much simpler. However, if the above instructions are observed, the deflecting wedge can be oriented in any desired course, limited only by the precision and accuracy of the drill-hole survey.

Three pin holes are located in the top of the deflecting wedge (Fig 2). The deflecting wedge is pinned to a half section of drill rod (Fig 5) with one iron and two copper or brass rivets. This may be varied to all copper or brass rivets, or to as many as two or three iron rivets. The practice is related to the judgment of the operator. The deflecting-wedge assembly must not shear before bottom is reached. On the other hand, if too much pressure is required to shear the rivets, the deflecting wedge may be deformed. After shearing the wedge, the runner should rotate the rods slowly before pulling. This precludes lifting the wedge out of the step.

We consider it good practice to put a slight convex sweep or bow in the thin upper part of the deflecting wedge.* The purpose of this is to assure that the thin top of the wedge will hug the wall of the hole to prevent the wedging bit from catching on it when lowered onto the wedge. If this procedure becomes standardized with the manufacturer, the wedges may be made up at the shop with the sweep added.

The overall length of the wedge assembly is measured carefully. The drop in the step is 8 in. The pilot wedge may seat into the drive wedge at once but, generally, the rods must be rotated slowly until a drop indicates that seating has occurred. When the wedge assembly has been properly seated in the drive wedge, the rivets are sheared with the hydraulic swivel head and the rods pulled from the hole.

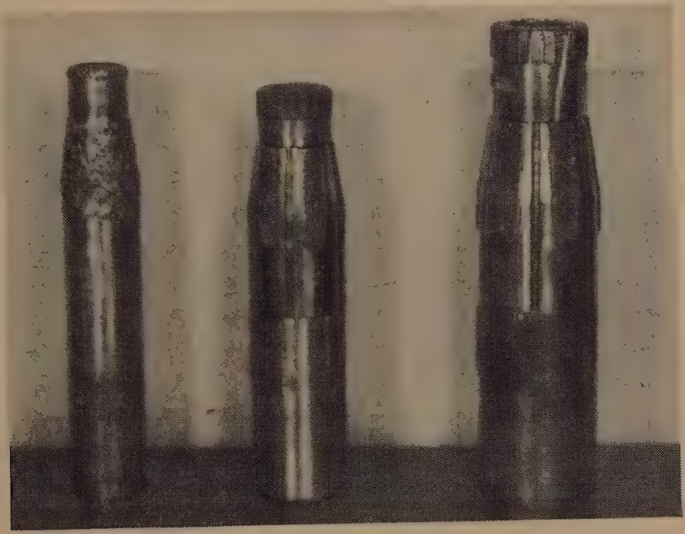


FIG 6—WEDGE BITS.

### *Drilling from the Wedge*

Not only is the Hall-Row technique a directional deflection, but it is one permitting a branch hole of the same size as the parent hole. The heart of the drilling equipment is a tapered reamer which the drill runners term a "wedge-bit" (Fig 6). For example, in wedging a *BX* hole, the tapered reamer is made to accommodate an *AX* bit in the forepart. A diamond-set tapered section increases from *AX* to *BX* size over a length of 3 in. The upper portion is threaded for both a *BX* and an *AX* single-tube reaming shell. Some styles also have an *AX* left-hand threaded box in case of the necessity of fishing.

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* Suggested by Mr. Frank Picard.

Our first wedged hole (*BX* size) was drilled with a 10 ft *BX* single-tube core barrel which followed the *AX* bit, tapered reamer, and *BX* reaming shell. Later, we used a 5 ft *BX* single-tube core barrel and, still later, a 20 in. *BX* single-tube barrel. Our reasoning was that the shorter the chord of a rigid member, the greater the deflection. At present we favor a 20 in. single-tube core barrel.

The actual angle of our deflecting wedge is 1°50′. This is virtually 2° angular deflection, especially since our tests are read to the nearest half degree of angle. The branch hole is tested for deviation at a point from 30 to 50 ft past the wedge. We have experienced all values of deflection from the theoretical 1°50′(2°), to virtually no deflection. The average of 21 wedges has been 1°6′ deflection. Several important factors are: the nature of the ground in its uniformity, toughness and degree of fracturing; the attitude of the bedding, stratification or foliation; and the relative deviation of the hole previous to the first wedge.

Another possibility that was investigated was the drilling of a smaller size hole after leaving the wedge in order to obtain maximum deflection. This hole was reamed to full size after some 30 ft of drilling past the wedge. It is necessary to balance the gain in deflection with the additional cost of reaming. This practice is followed now when only half or less of the theoretical deflection is obtained.

If the wedge is being placed to by-pass an obstruction, little attention is given to the character of the ground at the point of deflection. On the other hand, when a branch hole is planned some distance above the bottom of the parent hole, considerable discretion should be exercised in the choice of ground in which to set the wedge. Formerly the prevalent belief of the operators was that "soft" ground was better to ensure ease of drilling and maximum deflection.[3] The experience of Mr. Frank Picard* led him to observe that dense to firm ground is superior because the hole is more likely to be approximately the size drilled. He found that a hole in "softer" ground commonly is somewhat oversize and not suited to setting a wedge in it. Our experience has collaborated this observation. In one *BX* hole we know that the diameter exceeded 18 in. Local conditions determine the relative merits of the ground and some experimentation may be required in each district. Application can also be made of a drill-hole caliper.

### *Spacing of Wedges*

When the deflecting technique is applied to secure maximum displacement from the parent hole, the greater the number of wedges that can be placed in the hole, the greater the displacement. We realize there is a minimum distance of spacing wedges beyond which the excessive strain results in impractical wear and breakage of equipment. We thought this minimum distance would be about 40 to 50 ft, but our average to date has been 65.5 ft. The minimum distance of spacing has been 39 ft, but in that instance the drill rods tended to bind and promoted a broken coupling. We recovered the rods and replaced them with new ones, after which no further difficulty was experienced.

We have used as many as six wedges in one hole, five in two holes, four in a fourth hole and lesser numbers in other holes. The distance between the first and sixth wedge was 386 ft. This hole was nearly vertical where the branch left the parent hole. The runners noted a considerable bind or resistance to turning when using five or six wedges. Wedging with a large number is more difficult when the parent hole is not vertical, perhaps less than 80° from the horizontal. The maximum horizontal offset experienced to date in our wedging has been 150 ft.

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* Personal communication.

Fig 7 is a vertical section through one of our surface drill holes showing the parent drill hole with two branch holes. The south-branch hole was drilled first and the north-branch hole was drilled later. The value of these wedged holes can be seen readily from the illustration when one considers how the ore in the parent hole might have been extended had the branch holes not been drilled. The branch hole to the south established the presence of a fault and the amount of displacement by faulting.[4]

### Summary of Modification

The general practice and procedure described in this paper comprises our current technique. The discussion would not be complete without mentioning the changes made to the Hall-Row equipment and operation. Some of the changes were anticipated before the equipment was used while others were developed as we acquired more experience.

1. Our first change was to omit the centering sleeve on the deflecting wedge by substituting rivets. This eliminated the use of the rose bit to drill away the collar. Another advantage is that it avoids having the metallic cuttings of the sleeve in the bottom of the hole. Metal slivers are a great hazard as they may jam the core barrel. We have designed a magnetic extractor which removes metallic particles that may be produced in drilling on the deflecting wedge. Our experimentation led to the combination of rivets mentioned previously.

2. Since all our surface drill equipment and some underground equipment have hydraulic swivel heads, it appears logical to shear the various members with the hydraulic rather than employ a sledge hammer on an old coupling.

3. The design of the drive and pilot wedges was modified. The lower bevel was added and the length of the flat face increased. The thought behind this was to prevent the deflecting wedge from canting in an oversize hole. The increase in the flat surfaces was to give more bearing surface in

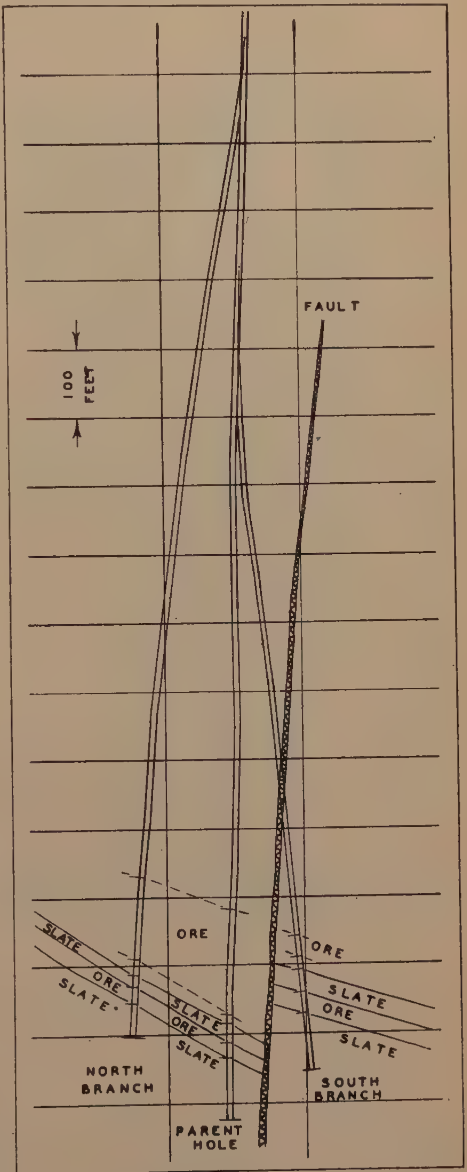

Fig 7—Vertical section through wedged hole.

case mud or caving material has partially filled the step (Fig 8). Actually, the design of the lower bevel may be academic because each hole probably has some mud to prevent good contact on these surfaces. Thus one must depend largely on the long vertical surfaces to prevent rotation, and the sweep of the deflecting wedge to prevent its canting into the hole.

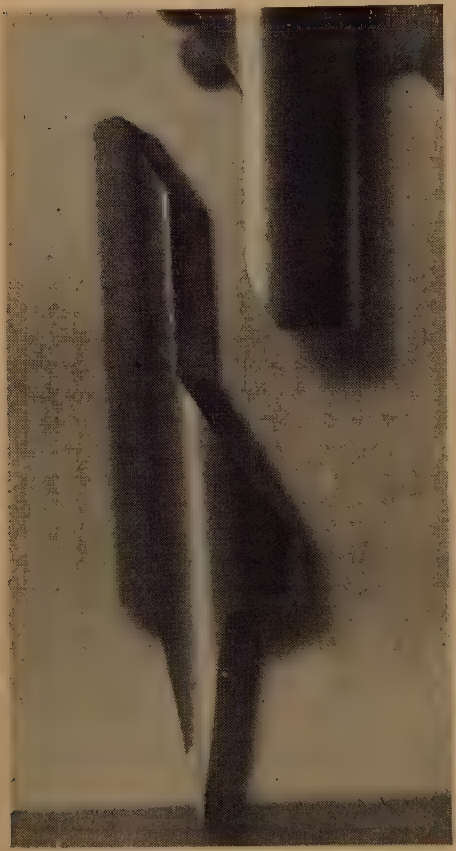

Fig 8—Drive and pilot wedges.

4. Early in our practice we found that there must be more solid round stock below the end of the concave section of the deflecting wedge and the pilot wedge. It is this extra section that creates a pillar between the parent hole and the branch hole. The pillar is vital to prevent forcing the pilot and deflecting wedges into the branch hole or having them cant into it, as occurred where the wedge was extracted in drilling by becoming lodged in the core-barrel. Later, we made an additional part which we called the "wedge extension." It is 5 ft long and one or more of these may be added if the operator suspects that the condition of the ground warrants an additional pillar.

5. After cutting several long slivers from deflecting wedges, we arranged the design of the wedge with the wings or upper edges milled at a bevel to its longitudinal axis. This eliminates the long thin "shoulders" at the upper end of the deflecting wedge.

6. Another possibility is that of cementing the wedge in place before drilling the branch hole. There may not be enough clearance in many holes, however, for the cement to be effective. We tried cementing one wedge assembly but had no conclusive results.

7. The receptacle for the glass vial in the *BX* and *AX* clinometers was increased to 1$\frac{3}{16}$ in. in diameter to accommodate the 1$\frac{1}{8}$-in. glass tube, a standard size in our operations. Wall thickness would not permit this change in the *EX* size. The balance of the changes mentioned above have been incorporated into a set of standards which applies to *BX*, *AX* and *EX* sizes. We keep on hand all the parts for all three sizes of wedge assemblies.

The changes in equipment and practice that are cited above are the result of the cooperative efforts of many persons. Mr. Ernest A. Allen and Mr. Edwin Jacka of The Cleveland-Cliffs Iron Co. deserve special commendation for their assistance. Messrs. Robert D. Longyear, Frank Picard and Morgan Beckman of the E. J. Longyear Co. were most cooperative and ready to collaborate on changes in manufacture and application.

## General Comments

### *Average Time of Operation*

A large drill crew was used when the first wedges were set so that our personnel could be trained to a standard procedure. Thereafter, the crew consisted of the runner and a helper. More time is expended in setting the first wedge for a branch hole. Since it is our practice to fill all surface holes with cement before abandonment, whether wedged or not, this time should not be charged to the wedged hole. An average of 10 man-hours is spent in placing the wooden plug. This includes washing out the hole and chopping out an occasional plug that does not reach the desired depth.

The average time involved in setting the drive wedge and testing with the clinometer is 8 man-hours. This is done by a 3-man crew, as a technician directs all drill-hole surveying. Setting and shearing the deflecting wedge requires 8 man-hours. Drilling off the wedge until a full diameter is attained in the branch hole takes about 12 man-hours. To the above should be added about 8 man-hours to prepare the wedge equipment on surface, to read the tests and to rivet the pilot wedge into the deflecting wedge. The total time will vary with the depth of hole and the general nature of the ground. We have set wedges at depths ranging from 1291 to 2376 ft. Our deepest hole on the Marquette Range to date is 3672 ft* and contained three wedges in the interval from 2186 to 2281 ft.

### *Average Cost*

Any cost analysis must consider the fixed and variable charges. The fixed charges must include the cost of the complete wedge assembly, the labor for something like 56 man-hours involved in the procedure, depreciation of equipment, power, diamond wear and footage drilled in each branch hole to make room for the assembly. The variable charges are controlled by the trouble that may arise.

### *Trouble*

The "middle name" of diamond drilling is "grief." A great many details may arise to beset the operations. A few of our difficulties have been:

1. Cement would not plug hole.
2. Wooden plug hung up in hole.
3. Drilled off a portion of the wedge.
4. Wedge extracted by core-barrel.
5. Wedge tipped and lodged across branch hole.
6. Drill rods broke at a coupling.
7. Caving ground required chopping and drilling.
8. Casing broke at joint.
9. Steel slivers jammed core-barrel.
10. Hole fouled with rod dope.
11. Deflecting wedge sheared before reaching drive wedge.

We have had the pillar between holes break loose and allow the drive wedge to fall to the bottom. This required drilling over the wedge to remove it.

We have lost two core-barrels in one hole from steel fragments which lodged along side barrels so tightly that they could not be jarred free.

### *Precautions*

1. Do not use bits set with black diamonds if trouble is suspected or anticipated. Their value is too high to be lost in the hole.

2. Avoid large amounts of rod dope or vibration grease.

3. Remove the hard ferrule from the top of the core barrels before using in a wedged hole.

4. Post in a prominent place the depth at which each wedge has been placed. It is imperative that the runner lower slowly past each wedge.

### *Conclusions*

Ground that drills well with little caving should be well adapted to wedging.

The modified Hall-Row wedge equipment is simple to operate.

A great saving may be made in reducing the number of surface setups by deflecting branch holes from a parent hole.

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* Since preparation of the paper, another hole has been drilled to 3752 ft.

It is possible to control the deflection of a wedge hole with this equipment.

## ACKNOWLEDGMENTS

The writer wishes also to acknowledge with thanks the encouragement and suggestions given him by E. L. Derby, Jr., Chief Geologist. Kind assistance was given by W. E. Bertholf, Jr. in the critical reading of the manuscript and by Mrs. Belle Bloch, Archibald Minnear and George Olson in preparing the manuscript.

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# Educating and Training Economic Geologists of the Future

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(New York Meeting, March 1947)

## Abstract

This paper discusses education and training for economic geologists other than petroleum geologists. Candidates enter economic geology through liberal arts colleges, engineering schools and university graduate study. Whatever their backgrounds, rigorous courses are desirable—now conceded in theory, but not adequately recognized in practice. Besides basic sciences (each studied long enough to give a fair comprehension of its basic tenets), a functional command of English and two modern foreign languages is needed, as well as a broad understanding of mining, milling and metallurgy. A year of economics and finance is recommended. Surveying and drafting are essential skills. Yet geology should be represented in each term; otherwise the initial enthusiasm for the science, once contact is broken, may never be regained.

A five-year curriculum is thus designed to take care of all major requirements and culminate in the M. S. degree. This curriculum reduces "cultural" courses below the average liberal arts requirements, but this is regarded as necessary. The pursuit of "culture" is assigned, as in most engineering and medical curricula, to extra college years or extracurricular time.

To avoid the danger of over-crowding the schooling, the Doctorate in geology is preferred wherever possible. Its two added years should permit wider professional education; more browsing in "cultural" pastures; due attention to broader social implications; and some training in the research now increasingly required of the mineral industry.

"In-course" training through cooperation between school and industry is especially advocated. The Institute should take the initiative in stimulating this practice.

## Introduction

Training a student in geology for a career in the mineral industry is still a moot question, one that is likely to be looked at from many different angles, each of which will reflect the background of the individual considering it. From what follows, the training of petroleum geologists is omitted since, although theirs is an equally important phase of economic geology, it is based upon different essentials. The general subject was recently effectively discussed, but from a somewhat different viewpoint, by Professor McKinstry.[1] The practical problems that confront the young mining geologist are highly varied and in the eyes of his employer apparently almost any demands seem justified. The mining executive wants from the mining geologist results related mainly to overall operations, reports or forecasts that are largely recordable on paper and chiefly in terms of "discovery thinking." The local mining engineer may have many specific questions of his own, chiefly relating to details of geology that apply to mining practice, which he will address to the mining geologist; these usually deal with production quantities; frequently also they necessitate the immediate handling of pressing but apparently minor matters: shall the drill be moved; how soon will we encounter the watercourse which made so much trouble for us above? The client employing a consultant often wants advice or good judg-

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Manuscript received at the office of the Institute March 16, 1947; revision July 28, 1947. Issued as TP 2278 in Mining Technology, November 1947.

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[1] References are at the end of the paper.

ment on the cost or return expectable in a given operation, and such advice is in part at least dependent on *financial* experience and wisdom. State and federal surveys ask still other questions, largely requiring a more purely scientific as distinguished from a technical and executive background; what is the rock sequence here, they say, or what is the local structure? These questions, typical of the variety of functions that the mining geologist may be called on to perform, may grade into each other, but the spectrum is a wide one and the questions run the entire gamut of geologic knowledge and its application to mineral production, fabrication, and economics.

The practising mining geologist, furthermore, may come out of four quite different kinds of backgrounds, or their combinations:

1. A *liberal arts college*, often differing in various cases to an almost incredible degree.
2. A *school of engineering*, especially of mining engineering, with a mining or geologic "option" as the nucleus.
3. A specialized *graduate training in geology*, broadly conceived.
4. *Specialized graduate training in mining or engineering geology*, narrower but deeper than the last.

It should also be remembered that individual schools show large variations within themselves and that even in a given department of geology agreement may be far from complete as to both objectives and methods.

Within the last year or so the National Research Council and the Geological Society of America have jointly sponsored a series of discussions of this subject.[3] In what follows reference will be made to suggestions arising from those discussions but special emphasis will be placed upon certain conclusions not hitherto mentioned.

### Need for Rigorous Training

Perhaps an analogy is justified. In the 50 years of medical education included in the present century striking changes are recorded. Medicine used to be a rather casual subject, either pursued by itself without any stipulated background or grafted on top of a training in entirely unrelated subjects. Instead, today, in the middle of the century, the medical curriculum calls for a fairly rigidly prescribed baccalaureate degree, followed by from two to four years of professional work in the field of medicine, internships, and even, in the best cases a Ph. D. in a special field. It is, in fact, similar in its rigidity to the engineering curriculum, but generally longer. Medical schools differ somewhat in the degree of their emphasis upon freedom of choice in the first two collegiate years. Above that level their curricula become more and more convergent as the courses near completion, and no medical school would dare to build upon a foundation that lacked basic courses in the sciences—physics, chemistry, mathematics, anatomy, pathology, bacteriology, physiology, and physiological chemistry. Then only and at last come medical practice, surgery, clinical diagnosis, and the like. Increasingly, medicine as a professional subject becomes essentially a postgraduate science, following essentially a bachelor's degree in the prerequisite subjects.

Training for economic geology is far from uniformly recognizing any prerequisite elements similar to those called for in medicine or engineering. A rigorous criticism, therefore, of the continued "flabby" attitude existing in some quarters toward geologic training is today entirely justifiable. But it will be misleading not to recognize the reasons for this attitude. It arises in large part from these facts:

1. Geology is approached in part as a "cultural" science offered in liberal arts colleges as a broadening subject, a course and curriculum not designed for the professional geologist. As thus presented it may serve another very important purpose: it may

be orientative—by opening before a student's eyes the possibility of a hitherto unrecognized career, whether in mining or in geology.

2. Few laymen who might enter geology and indeed few enough college administrators include geology among the obviously possible professions. It is but little taught in high schools, and its relation to the mineral industry is not well understood. Nor is the mineral industry itself generally regarded as basic in any except mining communities. Hence there is little public recognition of the need for rigorous professional training with a fixed minimal background.

3. All the various subdivisions in the broad professional field of geology are very divergent indeed, involving backgrounds that differ considerably. The greatest contrasts are between the biologic aspects of geology and the physico-chemical aspects. Those interested in one of these fields do not well understand the career in the other.

## BASIC ELEMENTS OF A DESIRABLE CURRICULUM

Despite such contrary influences, however, and largely because of the agitation started by certain thoughtful geophysicists and geologists seven or eight years ago, there seems now to be a general recognition, in theory if not in practice, that all geologists, including the economic geologists with whose education this paper deals, need basic training in mathematics, physics and chemistry. Less broadly recognized is the importance of basic training in biology, certain social sciences, languages and special engineering subjects.

Some factors make the plans included in these generally laudable reforms difficult to put into effective practice. Among these are the problem of fitting into the undergraduate curriculum all of the courses which seem desirable; the difficulty of achieving a widespread recognition of the fact that a biologic background is as badly needed for many phases of geology as is a sound training in mathematics, physics and chemistry for other phases; the lack of recognition that the social sciences, especially economics, are important elements for the mining geologist; and the failure to realize the importance of keeping the student who once wanted to be a geologist in early and constant touch with this main subject of his interest for the sake of morale. Still, the prime objectives of the proposed reforms were sound.

In the training of geologists for the mineral industry, two more kinds of subjects have been largely neglected in the past. The importance of English has been stressed because of its use in speaking effectively and in writing suitable reports. To all that has been said on this score a fervent "Amen" is certainly merited, provided there is genuine integration with the curriculum and a constant emphasis upon good writing, in place of the superficial, short course popularly supposed to cure poor language command. Perhaps a more lively sense of good lecturing among the teachers in engineering and geology courses might also help this cause a bit!

But *American* mining is becoming increasingly *international* mining. Should not the mining geologist be required to know well and to speak fluently at least two modern languages other than English, by preference Spanish and one of the following three—German, French, or Russian? The last three are valuable instruments for conversation and they furnish access to necessary scientific and technical literature as well; Spanish is less valuable for the latter purpose perhaps but is essential for any man working south of the Rio Grande. Profiting from our war experience, the teaching of both reading and speaking should be functional.

One other pragmatically important unit in the curriculum of the young mining geologist includes the purely technical aspects of his work, several of which are in the nature of skills. These are too frequently not introduced early enough nor, in many cases, in large enough doses. If the student comes up through an engineering curriculum he has these skills at hand to at least a moderate degree. If he has worked toward an E. M.—the nearest to an ideal *undergraduate* background for advanced work in mining geology—he will have acquired them. I refer especially to drafting and surveying (surface and underground, transit and plane-table). If the student graduates as a "pure" scientist, and his first employer asks him to do simple drafting or surveying, he may, because he cannot rise to *that* occasion, never be given the opportunity to demonstrate his command of *geology*, however outstanding that may be. Further, the student expecting to become a *mining* geologist should also know something about mining, milling and smelting. Some at-homeness with the lingo and meaning of these related subjects is the least that should be expected.

Finally, if there has been any general error in recent discussions centering around the proposed new curriculum for the embryo economic geologist, it has perhaps been the overemphasis upon and the proposed excessive introduction of the more "basic" sciences to the exclusion of geology, and the consequent starvation or at least malnutrition of the student's spirit and interest in the earth sciences. Most experienced planners and teachers of curricula in professional geology will agree that if geology is kept *entirely* out of the earlier college years of the schedule, the student ends by becoming a physicist, a chemist or an engineer; a modicum of his especially chosen subject is essential *each* term or quarter or semester: (1) to maintain his interest in it by showing its continued bearing on his work, and (2) to give him the confidence in its grasp that comes only with steady contact: five months at 20 hours a week in geology is nowhere nearly as good a training as 10 months at 10 hours a week, even though the total time is the same; in fact, the second consideration is why many excellent mining engineers are very poor geologists.

## DETAILS OF A PLANNED PROGRAM

With these principles in mind, and yet with the desire to avoid any departure from present teaching practice which might make a proposed plan hopelessly impossible of fulfillment, a rather fixed five-year program is outlined in Table 1. This schedule might well carry the candidate through the M. S. degree and prepare him as a fairly competent junior mining geologist. It is not a curriculum in engineering, but it can be adapted to both schools of engineering and colleges of liberal arts. The adjustment of hours is, of course, quite arbitrary, but the sequence is designed with some care so that the student will maintain relations with a given subject (e.g., chemistry, physics) through several years, and thus become "mellowed" a little in each according to the principle previously laid down. The courses could be given either in a mining school or a liberal arts college of moderate size, except that the liberal arts college is not always equipped to give the asterisked courses and certain other courses are not always provided by engineering schools.

The optional hours in the third and fifth years might perhaps be devoted to: strengthening the training in the languages; further courses in structural geology; special training in field or laboratory techniques, such as mine model construction; special work in mineral economics, or mining law; special techniques applied to mineralogy; advanced work in geophysics; or any purely "cultural" subject. The order of listing these optional subjects expresses the personal preference of the writer with the average student in mind; but others will doubtless recognize a different sequence of priority. Circumstances (for example, the background of the average entrant, the specialties of the individual institution by predisposition of staff or peculiarities in local facilities) will influence what should be done; allowance must obviously be made for such reasonable adaptation of the woof of the curriculum to fit this more or less fixed warp.

In some of the courses listed a serious difficulty is the lack of cooperation by teachers of chemistry, physics, languages, economics and the like, who present their subject, as though for specialists in their respective fields—a thing not unknown, by the way, among geology teachers when the shoe is on the other foot!

TABLE 1—*Five-year Plan*

| 1st Semester | | 2nd Semester | |
|---|---|---|---|
| First Year: | | | |
| Introductory geology | 4 | Introductory geology | 4 |
| Introductory chemistry | 4 | Introductory chemistry | 4 |
| Mathematics | 3 | Mathematics | 3 |
| Language | 3 | Language | 3 |
| * Drafting | 3 | * Drafting | 3 |
| Total | 17 | Total | 17 |
| Summer: * Surveying: Surface and underground | | | |
| Second Year: | | | |
| General mineralogy | 4 | Optical mineralogy | 4 |
| Descriptive geometry | 3 | Structural geology | 3 |
| Introductory physics | 4 | Introductory physics | 4 |
| Language | 3 | Language | 3 |
| Economics | 4 | Finance | 4 |
| Total | 18 | Total | 18 |
| Summer: Geologic field course | | | |
| Third Year: | | | |
| Physiography (?) | 3 | Sedimentation | 3 |
| Quantitative analysis | 4 | Quantitative analysis | 4 |
| Mathematics | 3 | Mathematics | 3 |
| * Elements of mining | 4 | * Elements of mining | 4 |
| Optional (language?) | 3 | Optional (language?) | 3 |
| Total | 17 | Total | 17 |
| Summer: Field experience | | | |
| Fourth Year: | | | |
| Physical chemistry | 3 | Physical chemistry | 3 |
| Advanced physics | 4 | Advanced physics | 4 |
| Petrology | 4 | Petrology | 4 |
| Elementary biology | 3 | Paleontology | 3 |
| * Ore dressing | 4 | * Introductory metallurgy | 4 |
| Total | 18 | Total | 18 |
| Bachelor's Degree Awarded | | | |
| Summer: Field experience | | | |
| Fifth Year: | | | |
| * Introductory geophysics | 4 | * Exploratory geophysics | 4 |
| Ore deposits | 4 | Non-metallic deposits | 4 |
| Ore petrology | 3 | Thesis | 3 |
| Advanced general geology | 3 | General examination in geology | 3 |
| Optional | 3 | Optional | 3 |
| Total | 17 | Total | 17 |
| Master's Degree Awarded | | | |

* Course not always given at liberal arts college.

## CULTURAL VERSUS PROFESSIONAL TRAINING

Objections will be raised to the effect that this training is too specialized, too narrowing, too professionalized. Or, because the small liberal arts college is the natural "source" of "culture" and is a most important cornerstone in our educational system and because the curriculum in many smaller liberal arts colleges is not adapted to such a specialized course as that outlined above, it will be asserted that this fundamental, essentially technical training cannot be effectively administered in such schools. This in turn opens the much broader question: what should be the function of the liberal arts college in American life?

In the writer's opinion the liberal arts college can serve three very useful purposes for the average American college student.

1. It should give young men and women a taste of many fields and aid them in selecting for themselves a specific and specialized career.

2. It may offer to the young specialist in the humanities and the social sciences a good start toward his selected career.

3. It should furnish a cultural background for later life for those who need it because such a broadening background was lacking in their earlier years at home or in the secondary schools.

These statements are based on several debatable but defensible premises: (1) that "culture" is absorbed through the skin, not taken by mouth (a thesis that I shall gladly defend if called upon to do so), (2) that for the sake of humanity we should not undertrain our professional geologists any more than we can afford to undertrain our professional engineers and physicians, and (3) that the best way to combine intensive professional training with "culture" (a valuable adjunct for any person if its acquisition does not prohibit the basic training essential to make a man professionally trustworthy) is to serve up the "culture" in the form of lectures, demonstrations, and the like, taken *without* credit in *spare* time, just as such interests have to be pursued by mature persons in later life.

The small liberal arts college certainly has a very important function to fulfill under all three headings mentioned above, but these functions are not necessarily closely related to the training of economic geologists and to convert them in part to such a purpose requires courage and thought. This problem of liberal arts training in geology has been well summarized elsewhere by Dr. Delo, himself a geologist with extensive experience as a teacher in a liberal arts college. Delo[3] and Lougee,[4] while rightly stressing the "cultural" use of geology showed that even in the liberal arts college a good beginning can be made toward professional training if great care is used in planning and not too much freedom of choice is left to the student. The greatest potential strength of such an institution is unfortunately also its potential weakness. There is little doubt that its teaching is generally more inspired than that at other institutions, for the great teacher finds here his most effective medium and his readiest personal contact with students. But curricula can also become very unbalanced, very myopic. There are at least two such institutions where economic geology, which should certainly be a "peak" subject coming late in the student's career and founded solidly on other studies, is given in the 2nd or 3rd year. One institution actually permits its students to study ore deposits before they study chemistry or mineralogy! The shallowness of such curricula is obvious. One can only ask, "Why"? with a shudder.

### FUNDAMENTAL CRITICISMS OF THE PROPOSED TYPE OF CURRICULUM

Quite aside from the details of courses and hours and the question of a "liberal" education, there are several more significant criticisms of the general type of curriculum outlined above, however. In one form or another, they turn on three points:

1. Is it wise to so "crowd" the student especially in his less mature years? Does he have the time needed to absorb and think about the subjects? Or does he become an unimaginative, routine worker in consequence of the habit-of-mind engendered by this constant pressure on his time, this constant preoccupation with a single objective.

2. Do the proposed courses make due

allowance for the student's need for developing his own specialized interest and bend within the broad field of geology? During the war a certain young geomorphologist was forced by the national emergency to turn to economic geology; he has been eminently successful and happy in this new career. But the danger involved in such cases, the possibility that the young man might not have found his niche, is a very real one.

3. Finally, does the graduate come out so specialized in training that if, rather than scientific and technical, great executive responsibilities of a broader, more human type are given him, he fails miserably?

### The Doctorate

A solution for these significant objections and, if the foundation has been adequate, the best training anyway for the mining geologist seems to be the Ph. D. degree, either in geology alone or in combination with such subjects as mining engineering. This statement follows an earnest effort to look into the future. Already a decade ago the rising demand for higher training put the average holder of a Bachelor's degree at a marked disadvantage in comparison with men who are Masters or Doctors in geology, though this was not clearly foreseen 25 years ago. This heightened demand for further training is not yet as well marked in the mining industry as in state and federal geological surveys, the petroleum industry, and teaching. But it is certain that, as American mining companies increase their foreign interests, they, like the petroleum companies before them, will call for more experienced and better trained men. This will be because of the great risk involved in sending geologists far afield whose limited training makes their success in a new, challenging environment only a "long shot."

Another and more urgent reason, however, will force upon the mineral industry this continued raising of educational qualifications for geological jobs. The intensified search for minerals, and especially for deeply hidden deposits, makes men with the proper *research* background more scarce on the one hand, more desirable on the other. Indeed, several of the larger mining companies in this country are now actually engaged in explorational research on an appreciable scale. If the trend continues for another two decades—and everything points to its doing so—the Ph. D., far from being regarded as a mere decoration, will become an essential. Let those now faced with the responsibility for advising the mining geologists of 1970 take heed!

A corollary arising out of this consideration is the importance of training in research techniques of a rather specialized nature. Aerial photographs are now used stereoscopically in geologic mapping. X-ray technique is proving valuable in mapping alteration halos around mineralized areas. Special geochemical training in trace-element detection is beginning to be useful. Though less broadly used by mining than by petroleum companies, geophysics continues to be of major interest as a tool for mineral exploration.[5,6] In quite another aspect of the work, economic problems related to corporation finance, to S. E. C. hearings, and to tariff policies are increasingly calling upon economic geologists. Now in the peace, just as in wartime though less urgently, estimates of national resources are in demand. Then there are the well recognized but still new and poorly known techniques of geology and mineralogy as applied to the study of ore composition, which affect search, production, and metal extraction of the rarer elements. To date mining companies, with a few noteworthy exceptions, have relied almost wholly on consultants for most of such basic work. Few mining companies have geophysicists on their staff; geochemists are unknown in the development field; and structural and areal geologists are certainly not yet widely employed as regular staff members. The wisdom of the practice by which large companies ask high-priced consultants to do much somewhat routine but specialized work, which might frequently be done equally well by regular staff members, is certainly questionable.

Very little of the types of investigation mentioned above can be adequately covered without specialized training at the level above the Master's degree because, as already implied, the candidate does not have enough time to study such special phases in the first four or five college years, particularly if he is to be broadly trained in the beginning. Only still higher studies can make it possible to follow thoroughly the many requisite subjects and to acquire the range of background needed when new problems present themselves. Most practising geologists can recall able men who came up from the ranks to outstanding roles though they lacked the M. S., the E. M., or even the B. S. degree. Such men were successful because they were exceptional, and the exceptional nature of their personal capacities is the reason why we cannot pick them off the trees. To make moderately certain of the adequacy of a man's training a higher degree will be necessary—an M. S. or even a Ph. D., just as the petroleum industry, the chemical industry, and fields of engineering broadly related to electronics have already recognized.

One final consideration favoring the Ph. D. is even more important. It is briefly that only the curriculum for the Doctorate affords the time for thoughtful, careful study; it alone affords study which does not have to watch the clock to see when the next class will begin. Very little careful thinking can be done on the time-table representing the normal undergraduate or early-graduate schedule, especially that in an engineering or "all-science" curriculum. For "exploration thinking" in the future, time must be afforded for contemplation, a practice that must begin during schooling. It might be added here that the Ph. D. is obviously most broadening if taken at another institution than that where earlier degrees were completed. This is especially true for men in geology, a profession greatly influenced by the *regional* outlook.

### IN-COURSE TRAINING

Even the best of curricula, the most carefully designed field and laboratory studies, the most informed and stimulating of faculties cannot give the student one essential element of his training—practical experience. For this we must rely on industry. "In-course training," widely accepted for mining personnel in Europe but hitherto practiced only slightly in any *systematic* fashion in this country, appears to be the best answer. It is advantageous to the student in broadening his experience; often it alone makes financially possible the continuance of his stay in college. It benefits the participating school by maintaining its outside contacts and by bringing concrete problems into the classroom, thus heightening the realism of the studies.

But "in-course training" helps the employer too. Through it he comes by temporary employment to know a group of promising young men in advance of the time when he has to commit himself to their acceptance or refusal as long-term employees. He is able at lesser expense than later to give them basic training for future work. As a corollary, a close liaison between employer and school may enable the careful checking of the employee's short-comings by both parties, the encouragement of promising young men to enter special openings in the mineral industry, while obvious misfits may be diverted into more suitable fields of activity, better for themselves and their employers.

The University of Cincinnati is one example, and Antioch College, Ohio, another, of the schools which have pioneered in working out a satisfactory solution of this problem. Some institutions only *think* they have: the writer was once assured by the Dean of Engineering of a good-sized state university that *all* his students found instructive summer employment; later an intimate acquaintance with many of those students proved how ill-informed the dean really was. The various state geological surveys and several federal services employing geologists have also pioneered in this field. It is to be hoped that the opportunity for work with the U. S. Geological Survey will continue to be regularly accorded to students under the new peacetime plan. Several of our state surveys have likewise been most helpful. On this score the Canadian Geological Survey has been our most energetic leader. Recently the U. S. Geological Survey has developed a very effective plan for apprenticeship for Latin Americans, available through the State Department.[7] Some ten young Latin Americans are now taking or are scheduled to take such training as regular junior members in field or laboratory projects. This is a wise and useful undertaking, perhaps even a case of enlightened selfishness.

Lest it be assumed that in-course training would, if made possible by industry, be pure idealism, attention should be called to realistic, closely designed, parallel plans for young engineers, reported upon favorably by several firms now engaged in such experiments, for example a large coal mining company and a great steel corporation.[8,9]

It remains to be seen whether the American Institute of Mining and Metallurgical Engineers has the vision to lend its influence, and the energy to set in train through its Mineral Industry Education Division a plan by which the better students in our mining schools, universities and colleges can obtain, both on the graduate and undergraduate levels, the in-course training which could be so valuable an adjunct to education and to industry.

## CONCLUSIONS

In brief then, the following statements in respect to the training of mining geologists seem justified:

1. A fairly rigorous but not necessarily rigid curriculum is needed, one which embodies the recognition that professional standards can and should be set up and maintained in this field.

2. Besides geology (the higher levels of which are frequently introduced too soon), basic elements in this training should include mathematics, chemistry, physics, biology, economics, command of two modern languages of world usage; elements of mining, milling and metallurgy; some skill in surveying and drafting.

3. The student should be kept in touch with geology throughout the curriculum, from the very beginning, so that his initial interest will not flag or be diverted into other fields.

4. Too many details should not be insisted upon in individual courses; courses should be broad, rather than long.[10]

5. The small liberal arts (undergraduate) college is the educational unit most difficultly integrated with this plan. It has a major function in spreading an understanding of geology and mining among laymen and persons who will never become professionally connected with the mining industry. It can also furnish the student with alternatives for choosing his career. Finally, such a college *can* train men for professional work of an advanced order *if* the program is carefully planned along pretechnical or technical lines.

6. An M. S. or an E. M. (if a five-year course) curriculum is useful for men planning to enter geology, but the standards are rising. This amount of training will soon not be sufficient to produce leaders. Even now it forces too rigid an attention to details and techniques.

7. The Ph. D. or its equivalent in length of training seems the most effective device

for assuring adequate coverage, sufficiently detailed training in each field, and enough time for the student to grasp the broader implications of his future career and to prepare for leadership.

8. Some type of systematic cooperation should be urged upon industry by the Institute, looking toward "in-course training" for our more promising students in mining geology.

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## DISCUSSION

R. V. COLLIGAN*—I can best illustrate my opinions by drafting another five year plan (Table 2). This plan is set up following the same stipulations outlined by Dr. Behre, namely: (1) the student is preparing himself for commercial work as a mining geologist; (2) he is attending an institution with a sufficiently broad curriculum (that is, engineering school and mining school as well as liberal arts college) to furnish the courses listed; and (3) he is not planning to continue his studies through to a doctor's degree before going into commercial work.

A comparison of my study plan (Table 2) and that submitted by Dr. Behre (Table 1) will at once point up the basic similarity, due to the obvious fact that there are so many essential courses which must be included in any outline of study. However, a more detailed analysis will show up certain differences, which, in fact, are only certain refinements on Dr. Behre's program. These I plan to discuss in order of their importance.

The first and most important change is the addition of 12 hr of optional subjects, making a total of 24 hr of optional subjects as against a total of 12 hr under Dr. Behre's schedule. This of course entails the elimination of certain courses, which I shall discuss later. However, I wish to make it clear that I suggest eliminating these courses not because I feel that they are unnecessary, but rather because I feel the optional courses are more necessary to the complete education of the individual.

I appreciate that Dr. Behre recognizes the importance of the cultural side of education, but I take issue with his thesis that the student will get it without credit and in his spare time. I do not think that the average college student, and certainly not the college freshman, is mature enough to know what he needs and where to get it in his spare time. I think it must be "spoon-fed"; he must be oriented through specific courses. University life (or should I say "college life") is too full of athletics, fraternity activities, dances, "bull sessions," and countless other enjoyable distractions, all of which contribute to the "full man." I fail to see how the average young American male can possibly sandwich in a necessary complement of "cultural activities" in his spare time. Since these latter activities do not fall within the pattern set by the "group," and show less promise of immediate return in enjoyment, or increased social status on the campus, they are bound to suffer in competition with the more popular, "group" activities.

No doubt much of the criticism levelled against arts colleges, namely, that the cultural courses are useless, is well taken, for the culture is not as carefully planned nor properly balanced as are the basic science courses for engineers, doctors, and others. There are courses in Elizabethan literature, history of the ancient world, modern English drama, American poetry, sociology, and so on—a veritable maze through which a student may wander, his decisions dependent in many cases on the question, "How does it fit my daily schedule?" rather than, "How will it fit into my educational pattern?" This pit-fall is especially threatening to the science major who has a few hours free for an elective course.

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* Freeport Sulphur Co., New Orleans, La.

TABLE 2—*Suggested Plan of Study for Economic Geologist*

| 1st Semester | | 2nd Semester | |
|---|---|---|---|
| First Year: | | | |
| Introductory geology | 4 | Introductory geology | 4 |
| Introductory chemistry | 4 | Introductory chemistry | 4 |
| Mathematics (Trigonometry) | 3 | Mathematics (Analytical geometry) | 3 |
| Language | 3 | Language | 3 |
| Optional (Survey course) | 3 | Optional (Survey course) | 3 |
| | 17 | | 17 |
| Summer: Surveying: Surface and underground | | | |
| Second Year: | | | |
| General mineralogy | 4 | Structural geology (Laboratory) | 4 |
| Descriptive geometry | 3 | Drafting | 3 |
| Introductory physics | 4 | Introductory physics | 4 |
| Language | 3 | Language | 3 |
| Optional (Survey course) | 3 | Optional (Survey course) | 3 |
| | 17 | | 17 |
| Summer: Geologic field course | | | |
| Third Year: | | | |
| Optical mineralogy | 4 | Sedimentation | 3 |
| Quantitative analysis | 4 | Quantitative analysis | 4 |
| Mathematics | 3 | Mathematics | 3 |
| Elements of mining | 3 | Elements of mining | 3 |
| Optional (Survey course) | 3 | Optional (Survey course) | 3 |
| | 17 | | 16 |
| Summer: Field experience | | | |
| Fourth Year: | | | |
| Advanced physics | 4 | Advanced physics | 4 |
| Petrology | 4 | Petrology | 4 |
| Economics | 3 | Finance | 3 |
| Ore dressing | 4 | Introductory metallurgy | 4 |
| Paleontology | 3 | Advanced structural geology | 3 |
| | 18 | | 18 |
| Bachelor's Degree Awarded | | | |
| Summer: Field experience | | | |
| Fifth Year: | | | |
| Introductory geophysics | 4 | Exploratory geophysics | 4 |
| Ore deposits | 4 | Nonmetallic deposits | 4 |
| Ore petrology | 3 | Geological problems (Laboratory) | 3 |
| Advanced general geology | 3 | Thesis | 3 |
| Optional | 3 | Optional | 3 |
| | 17 | | 17 |
| Master's Degree Awarded | | | |

I have in mind for the cultural courses the type of planned survey courses given at Columbia College, specifically the two year course in "Contemporary Civilization" and the one year course in "Humanities." It is my understanding that this form of courses is being adopted by more and more of the liberal arts colleges to satisfy just the needs I have mentioned here, the cultural education of the science majors. Although these are of necessity rapid surveys, nevertheless they give a student a complete, well-balanced picture of the history of the world and the men, the "thinkers" as well as the "doers," who have shaped it. They build the complete skeleton and form the broad pattern, upon which the individual student may enlarge at his leisure during his college days and, more particularly, in later life. They show the relation of the individual of our time to the world now, and how that relation has developed. That, in essence, is what I mean by culture—an understanding of the making of the modern man. I think that this is of vital importance to the appreciation of our daily life, and a man who graduates from college without receiving (or at least having been subjected to) this knowledge has been cheated. I do not think he can get it by a casual attendance to extra-curricular cultural activities, nor by a haphazard choice of electives in specialized cultural courses to fill in what free course hours he can spare from his major.

Now that I have stated my general thesis, I will list the particular variations from Dr. Behre's schedule.

1. Elimination of physical chemistry. I think two years of college chemistry should be sufficient.

2. Elimination of physiography. I think the introductory course covers most of the ground of the usual physiography course, especially insofar as land forms are concerned, and structural geology courses usually supply the necessary refinements.

3. Elimination of biology. If a man is going into mining, I fail to see the use of this course, except as an aid to more fully appreciating the paleontology course. The introductory phases of the paleontology course should cover the ground of a biology course as far as the mining geologist is concerned.

4. Reduction of drafting course from one year to one semester. A man should be able to learn the proper use of drafting tools and the necessary techniques of drafting in one semester. After all, this is a "finger-tip" technique which can be acquired by practice and self-discipline. Many of the refinements can be picked up in geological field work in the preparation of maps. Strict requirements on neatness and clearness of problem work by the geology department would, I think, do more good than that extra semester.

5. Optical mineralogy is moved from the second semester of the second year to the first semester of the third year, not only because it fits the proposed program better, mechanically, but also because I feel that the student should have completed his introductory physics before taking optical mineralogy.

6. A course in advanced structural geology is listed in the second semester of the fourth year. This could be a laboratory course on problems, or a discussion group with students assigned papers to review, or a combination.

7. Surveying: Surface and underground is listed, just as in Dr. Behre's schedule, at the close of the first year. I would like to add that this should be a required course, with transit work as well as plane table work.

8. It is suggested that the optional courses available in the fifth year be devoted to advanced geological work, or some particular phase of mining evaluation, mining law, rather than to cultural subjects. By the time the graduate year has been reached, we can hope that our hypothetical student has a sufficiently well-developed cultural background and will have retired from "campus life," so that he may utilize his spare time in cultural activities.

EVAN JUST*—I should like to endorse emphatically Dr. Behre's recommendations for in-course training. To my mind, this type of training is a must, both for geological and engineering education, if mineral education is to keep abreast of the growth of mineral industry problems.

Mining equipment is now so diverse, elaborate, and expensive that it is impossible for educational institutions to give adequate working familiarity with equipment in their laboratories. The place for this is obviously where the equipment is in use.

I am sure that most geological graduates who have not had practical experience feel more or less at sea upon entering industry. The realism which early acquaintance with the practical side would afford would add greatly to self-confidence and to the student's ability to profit from his instruction. Realistic knowledge on the part of the student would also operate as an important prod to those educators who do not maintain active contact with the industry and thus impart detached, academic knowledge to students.

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